THE MINER'S HANDBOOK

JOHN MILNE, F.R.S.

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THE

MINER'S HANDBOOK

A HANDY BOOK OF REFERENCE

ON THE SUBJECTS OF

MINERAL DEPOSITS, MINING OPERATIONS, ORE DRESSING, ETC.

FOR THE USE OF STUDENTS AND OTHERS INTERESTED IN MINING MATTERS

COMPILED BY

JOHN MILNE, F.R.S.

PROFESSOR OF MINING IN THE IMPERIAL UNIVERSITY OF JAPAN

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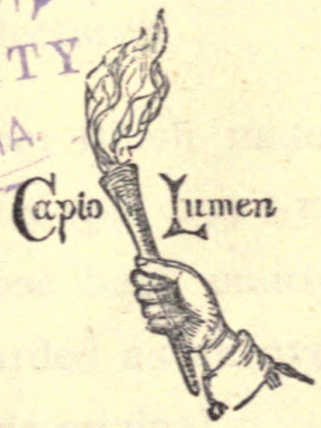

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PREFACE.

THE MINER'S HANDBOOK is a reprint, with corrections and additions, of a compilation, the first part of which was printed fourteen years ago at the Imperial College of Engineering, Tokio, Japan. Although here and there a few notes collected by the writer when visiting mining districts in various parts of the world have been added, the book as a whole is little more than a collection of well-known facts, put together in an order following the writer's lectures.

A list of works from which material has been derived is added as an Appendix (pp. 295—300); and if acknowledgement has in any case been omitted, it is hoped that the omission will be regarded as an oversight. But little, if anything, is put forward as original.

The object of the work is to enable students to remember more detailed descriptions of machines and processes, so as to allow them time to sketch from diagrams and models, and also

to serve as a book of reference for those who have some knowledge of mining matters.

For permission to use certain formulæ, and for suggestions and assistance when revising proofs, the writer offers sincere thanks to Mr. J. H. Merivale, Mr. W. Frechville, Mr. L. J. Healing, the Publishers, and above all to his colleagues.

TOKIO, *February*, 1893.

P.S.—It may be noted that, with the exception of the Title-page, Preface, and Contents, the work has been printed in Japan under the Author's direction.

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MINERAL DEPOSITS.

Classification of Mineral Deposits.—The following table illustrating the more general forms under which minerals are found in nature, has been drawn up from Von Cotta's Treatise on Ore deposits.

A. REGULAR DEPOSITS.

I. **Lodes or Veins.**

1. True Veins, transverse to the stratification of the beds in which they occur.
2. Bedded Veins, parallel to the stratification.
3. Contact Veins, at the contact of two dissimilar formations.

Belonging to these three divisions but named according to some peculiarity of their form or position are "gash" veins, lenticular veins, cross courses, cross veins or flucans &c.

II. **Beds and Layers.**—Mineral deposits which are members of a stratified series.

B. IRREGULAR DEPOSITS.

I. **Segregations** or deposits which are irregular in form but have definite limits.

II. **Impregnations** or irregular deposits without definite limits.

Another classification for Irregular Deposits is as follows:—

1. Deposits at the contact of dissimilar rocks.
2. „ embedded in igneous rocks.
3. „ „ „ stratified or metamorphic rock.
4. „ occurring in limestone.

A. REGULAR DEPOSITS.

I. LODES AND VEINS.

Lodes and their formation.

When we speak of a lode or vein we usually mean a fissure in the rocky crust of the earth which is filled with mineral matter.

In Australia a vein is called a *reef* and in California a *ledge*. Other equivalents are, *Gang* (German), *Filon* (French), *Filoni* (Italian).

A Champion lode is the main lode in a district. Running from it there may be ribs, strings, branches, scrins. Rake veins are inclined veins. Right running lodes are those following the general direction of lodes in a district; cross courses are those at right angles to this direction and counter lodes those which are oblique. The material in cross courses and counter lodes is usually different to that in the right running lodes. The sides of a lode are known as its cheeks. The more usual expressions are upper or hanging wall, and lower or footwall. The rocks through which a lode runs are spoken of as the country. A parting, as for instance, a film of clay between a vein and the country rock is called a selvage.

These fissures have been produced by various causes. The general contraction of a cooling globe is perhaps the most general hypothesis. When we feel the shock of an earthquake it may indicate that either a new fault is being formed or that an old one is being extended.

If the forming cause, whatever it may be, is evenly distributed then the fissures which result must evidently be parallel to one another unless there is an utter want of homogeneity in the rocky crust. And if we see in any district that the lodes are similar and that rocks are like in character, we can imagine the forming causes to have originated at considerable distance. We also see that two or more series of such fissures may be formed, those of each series being parallel to each other, but the members of one series making an angle with those of another series.

These different sets of fissures may have been produced at the same time, or more probably perhaps at different times as is indicated by the differences in character of the contained minerals,—a fact which is more or less pronounced in several mining districts.

If we regard these lodes as fissures in the earth which have been filled with mineral matter since their opening we must at once be

struck with the idea that it is highly improbable and almost impossible that such long gaping fissures as many of these lodes must have once presented could by any means have remained open without support.

It would seem that they must always have been full of liquid material at a great pressure or else they would have fallen in. If this were not the case it remains for us to suggest the way in which the long lodes we observe in nature could have originated. The suggestion as given by Fox, is that since the commencement when the fissures were of small size, they have gradually been increasing in length by some expansive action of the material they contained. This expansion may have taken place as follows. Because a lode from the metallic matter it contains is a better conductor of heat than the walls which bound it, and is connected with the hotter portion of the earth it will be subjected to comparatively great expansions and contractions. Another suggestion is that a lode may have gradually been formed by the accumulation and crystallization of mineral matter in a primary fissure or joint. If we regard a lode in this way we see the probability of its continunally tending to open itself and extend in length, evidence of which we have in the nature and appearance of the matter in many lodes, which seem to indicate that since their first deposit there have been many reopenings.

It might be expected that if matter is trying to force its way upwards from below it will rather tend to extend an old lode than to make a new one.

Length of Lodes.—Lodes vary considerably in length, the longest lodes being generally those which are broadest.

In Cornwall the average length of a lode is about a mile, in Freiberg three to four miles. In California there are lodes from 40 to 50 miles long.

The following are examples of the lengths to which lodes have been traced.

Wales (Cardigan and Monmouth)		6 to 9 miles.
Freiberg	Noble quartz formation	1 ,,
	Pyritic lead ,, ,,	.62 ,,
	Noble ,, ,,	1.24 ,,
	Baritic ,, ,,	.52 ,,
Hartz Mts		8–10 ,,
Schemnitz		2–4½ ,,

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The Old lode in the United Mines (Camborne) has been traced about 7 miles, but in this case, as in the case of nearly all lodes whose length is supposed to be great, it seems likely that mistakes may have arisen by confusing two or three lodes in nearly the same "run" and measuring them as if they were identical.

Strike.—Among the more ordinary questions put by a visitor to the manager of a mine about a lode, is an enquiry about its direction or *Strike*. On careful examination it will generally be found that this is not usually very accurately represented by the straight lines which are so often seen upon the maps of mineral districts. For not only may it vary in the position of its outcrop through its having been cut away by the erosion of valleys, but also it may often have irregularities of its own, and a slightly waving sinuous line following a general direction may be found to be more accurate that the straight one.

As before explained lodes are sometimes classified according to their direction as right running lodes, counter lodes &c. If the rocks are regular in a district as in the gneiss at Freiberg the lodes may be regular, but if the rocks are foliated and not homogeneous as in the schists at Andreasburg they may undulate.

In Cornwall the general direction of strike is ENE to WSW. In the north of England the right running lodes are E and W. In Prussia some of the iron lodes strike N and S. In a given district as at Freiberg the lodes may run in various direction.

When a lode changes its strike or dip it is said to come out of its hour.

Width.—In Cornwall the average width of a lode is 3 or 4 feet but we have examples of lodes like the Comstock in Nevada which in parts is 200 to 300 feet, or even more in width.

If we make measurements of any particular lode we shall generally find that it has considerable variations either as we descend in depth or as we trace it along its outcrop. This may be due to the rock at the time of its formation having yielded more at one point than at another. Another explanation might be that the first fissure had been a sinuous line and the rocks on one side of this had subsequently been moved relatively to those on the other side. Or lastly we might be able to shew that erosive powers either chemical or mechanical, had acted in one portion of the fissure more than in another.

In making any measurements of the width of a lode we must be careful to see that these are made at right angles to the walls. Sometimes however the lode "melts" into the country, in which case it is difficult to define the position of the walls.

If a lode has been opened by the rending action referred to when speaking of the length of lodes, we might expect that a lode passing through rocks of different characters would at different points along its length have been differently affected,—an action about which something might be learnt by observing if there is any connection existing between the width of a lode and the nature of the rocks it penetrates.

In Cornwall, Henwood observed that lodes were wider in slate than in granite. The greatest width was observed where tin and copper occur together. Copper lodes are wider than tin lodes. The lodes are wider within 100 fms. of the surface than at greater depths. Generally the width of a lode will vary with the hardness and other characters of the rock through which it runs. It will also vary with the nature of its own contents. In Cornwall there are lodes 10 ft. wide. The Great Devon Consols 42 ft. The Laxey Mine 12–18 ft. The Foxdale 36 ft. At Schemnitz there are lodes 120 ft. wide. At Kremnitz 30–90 ft. wide The gold-tellurium veins in Transylvania are a few millimeters in thickness. The silver veins at Kongsberg are only 2 or 3 in. wide.

Dip, hade or underlie.—The dip of a lode may vary as you descend in depth or at various points along the strike. The dip is spoken of as being steep or flat. In Cornwall it has been remarked that the north south veins dip more steeply than those running east and west. When a lode, changes its dip suddenly it is said to "throw a hook" if it increases in dip it "throws itself" while if the dip decreases it "raises itself." The Samson lode at Andreasburg dips in two directions while the Abendstern Morgen lode at Freiberg dips oppositely at its two ends. It has been observed that some lodes are richer in their steeper portions. At Monte Catini the copper ore chiefly occurred on the foot wall side of the lode.

In Cornwall it is said that if a lode dips beneath a hill, the superincumbent mass squeezes out the ore, and it becomes poorer.

Depth.—With the exception of veins like "gash" veins which are wedge shaped openings from the surface, the depth to which veins extend downwards has not yet found a limit. As examples of gash

veins certain lead deposits in the Mississippi valley may be mentioned If veins are openings from the surface downwards as we generally suppose them to be, their depth will probably be less than that of any veins which may have been formed by fissures opened from below. It is possible perhaps that veins of this latter kind may be represented by one or two doubtful examples which are said not to extend up to the surface of the ground.

The greatest depth to which a lode has been followed downwards is at Przibram in Bohemia, where one of the mines which is still being deepened has a depth of more than 1000 meters.

In July 1880 the Dolcoath mine was 410 fms. deep, Tresavean 340 fms.

The following are examples of the depth to which shafts have reached.

Clausthal	300 fms.
Freiberg	230 „
Schemnitz	220 „

The Samson mine at Andreasburg 410 fms. At Kongsberg veins 2 and 3 in. wide have been worked down to 266 fms. Comstock lode 500 fms. (See Shafts.)

With a change of rock a vein may appear to stop, For example the lead lodes of Derbyshire apparently cease when they meet the basalt or toadstone.

At Freiberg some lodes become poor with depth,—others however become richer. Certain silver mines in Bolivia became poor with depth. The gold veins in the crystalline slates of the Salzburger Alps are worked at the top of the mountains 6-8000 ft. high, while in the valleys, 2000 ft. below the lodes are poor.

In N and S Carolina gold is chiefly found in the upper part of veins. At a mine in Schemnitz, gold was also only found near the surface.

In California and Australia the influence of depth is not noticeable while in many mines, as the silver lead mines at Przibram, there has been a marked increase in the value of the ores with an increase in depth.

Contents of lodes.—In examining a lode we must observe the nature of the various minerals it contains and the proportions which these hold to each other. Sometimes we may observe that certain groups of minerals are often to be seen together, the presence of one

being favourable to the existence of the other. At other times the reverse will be remarked, the existence of one mineral being a sign of the absence of another.

The practical advantages to be derived from a series of observations shewing such results, are too obvious to be overlooked.

The following table shewing the association of ore in metalliferous veins is given by Phillips and von Cotta.

Two Members.	Three Members.	Four or more Members.
Galena, blende.	Galena, blende, iron pyrites (silver ores).	Galena, blende, iron pyrites, quartz *and* spathic iron, dial; logite, brown spar, calc spar- *or* heavy spar.
Iron pyrites, chalcopyrite.	Iron pyrites chalcopyrite, quartz, (copper ores.)	Iron pyrites, cholcopyrite, galena, blende; *and* spathic iron, diallogite, brown spar, calc spar; *or* heavy spar.
Gold, quartz.	Gold, quartz, iron pyrites.	Gold, quartz, iron pyrites, galena, blende *and* spathic iron, diallogite, brown spar, calc spar; *or* heavy spar.
Cobalt and nickel ores.	Cobalt and nickel ores, and iron pyrites.	Cobalt and nickel ores, iron pyrites; *and* galena, blende quartz, spathic iron ore, diallogite, brown spar; calc spar; *or* heavy spar.
Tin ore, wolfram.	Tin ore, wolfram, quartz.	Tin ore, wolfram, quartz, mica, tourmaline, topaz etc.
Gold, tellurium.	Gold, tellurium tetrahedrite (various tellurium ores.)	Gold, tellurium, tetrahedrite, quartz; *and* brown spar; or calc spar.
Cinnabar, tetrahedrite.	Cinnabar, tetrahedrite pyrites. (various ores of quicksilver.)	Cinnabar, tellurium, tetrahedrite, pyrites, quartz; *and* spathic iron, diallogite, brown spar, calc spar; *or* heavy spar.
Magnetite, chlorite.	Magnetite, chlorite, garnet.	Magnetite, chlorite, garnet, pyroxene, hornblende, pyrites etc.

Arrangement of material in a lode.—As the material in a lode may be collected together in so many different manners, for example it may be banded, in combs, ribbon like concentric, brecciated, compact, vughy, granular, disseminated, amygdaloidal &c., it is well that the arrangement should be observed because the structure of a lode will almost always tell us something about its history.

Sometimes we find the ore throughout the whole length of a lode to be tolerably well distributed with the gangue. Generally however it occurs more at one point than at another, existing in aggregations known technically by such names as *bunches*, *nests*, *chimneys*, *shoots*, *pipes*, *courses*, *squats*, *&c.* By carefully observing all the circumstances attending such deposits, rules and reasons for material aggregating more at one point than at another may perhaps be discovered. Closely connected with this is the fact that the mineral in a lode is often collected together in more or less horizontal *zones*. This is sometimes so marked that from time to time it may change the whole character of a mine, for example a mine which was commenced as a tin mine may at a lower depth yield nothing but copper, which in turn as in the case of the Dolcoath mine in Cornwall may eventually be replaced by tin. Some of the tin mines in the Erzgebirge with increasing depth yielded argentiferous copper.

A *pocket* or *bonanza* in the Comstock lode measured in length 650 ft., in depth 500 ft. and in thickness from 10 to 100 ft.

In some of the Freiberg lodes from 16 to 20 bands of quartz galena, blende, and pyrites may be counted. These bands may be arranged without any symmetry or in such a manner that similar bands occur to the right and left of one or more given lines.

In some lodes we find pebbles and even fossils.

Relation between the direction of a lode and its contents.—In certain districts it has been shewn that there is a great similarity in the contents of those lodes which run in similar directions. Thus in the Freiberg district where within an area of about 10 by 6 miles there are 900 lodes, we have the lodes classified into groups according to their direction,—the members of each group being characterized by the minerals they contain. These groups are as follows.

Stehende gänge run between N and NE, and contain silver, galena, and quartz.

Morgen gänge run between NE and E, and contain silver, barytes, and calcite.

Spat gänge run between E and SE, and contain silver and copper ore, with barytes in quantity.

Flache gänge run between SE and S, and contain little silver.

In Cornwall the ENE lodes yield tin, copper and sometimes lead, while the NS lodes usually yield clay and lead.

There are however exceptions to the above rules, thus at Clausthal, lodes in the same direction vary along their length, thus towards the north, spathic iron decreases, quartz and blende increase, barytes is rare and the galena is not so argentiferous. Towards the south barytes and quartz becomes less, blende dissapears and the galena is argentiferous.

In the Isle of Man we have the Laxey lodes running E and W and the Foxdale N and S, but the galena they yield is equally argentiferous. The NS lodes may however be more bunchy and unequal.

In Flintshire the galena in the E W lodes is laminated while in the cross lodes it is more earthy. The E W lodes also contain a little blende.

In the Duchy of Nassau and Siegen the N S and the EW lodes yield lead and iron equally.

Relation of lodes to each other.—In Cornwall it has been remarked by the miners, that if there is a certain portion of a lode which is rich, it is not unlikely that opposite to this point the portions of neighbouring lodes which are parallel will also be rich. This phenomenon is expressed by saying that "ore lies against ore." In order to understand this, we can imagine that at the time of the formation of these lodes parallel ruptures were formed along lines transversely to which the rockshad similar characters,—where each of the lodes crossed rocks of like character it is probable that the conditions accompanying the precipitation of mineral matter would be identical.

If two lodes intersect we may have them either simply crossing, or we may have one of them acting as a dislocator. In the former case at the point of crossing the arrangement of the materials of the two lodes, may give evidence of their having been filled simultaneously, or else we may have evidence that one of them was first filled, and was afterwards intersected by the second which was filled subsequently.

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In this latter case, and in the case where one of the lodes acts as a dislocator, we are able to determine the relative ages of the two lodes.

If two lodes intersect it has been observed that they often have an influence upon each other. At the junction of two lodes, minerals different to those found in other portions of the mine, together with a richer yield of ore, may be met with. These intersections will make certain angles with each other. These angles might be observed, because in some districts the obtuseness of the intersection is considered as having an influence on the richness or poorness of the mineral at the junction. In Cornwall it has been observed that at a junction of lodes having opposite direction of dip the contents are poor, while if they meet at an acute angle the contents are rich. Whilst enquiring about the intersections of lodes with each other, the existence of strings or small veins of ore connected with the main lode, might also be considered, as these also often give rise to accumulations of ore at their points of junction, which are not found in other portions of the lode. In some mining localities the miners hold that if these come in on the "hanging wall" side of a lode, they will be good for the lode, but if they leave it from the "foot wall" side they will "rob" it of its ore.

Relation of the contents of a lode to the country.— When examining a lode we must also examine the rocks through which it runs. Generally speaking we shall find that in most mining districts these are geologically very old. However lodes are not entirely confined to old rocks, as for example the copper lodes of Ani, in the province Ugo in North Japan, partly run through tuffs of Tertiary age.

For a district, or for a small country, it may be found that rocks of a particular horizon will carry a particular set of minerals, but this we shall see is probably due to the lithological character of the rocks of that area and the alterations to which they have been subject, rather than to their age.

However it is a well recognized fact, that certain rocks seem to be favourable to particular minerals. Thus tin is very often found in the vicinity of granitic rocks, whilst copper is more commonly associated with serpentines, diorites, chloritic schists and slates.

Granite yields gold, silver, copper, lead, antimony, iron, tin. Schists, mica slate and clay slate, yield a variety of minerals like gold, tin, copper, iron &c. Limestone yields lead in many countries. In Chili and Australia it yields lead and silver. In Belgium it yields zinc carbonate and in England iron and lead.

Observations of this kind are especially marked when considered in reference to some particular district, as for example in Cornwall where we have large bosses of granite penetrating through clay slate. In the former rock we generally find veins of tin, near the junction copper, whilst in the latter, at some distance from the granite are found the veins of lead.

Not only does the change in character of the rock effect a change in the nature of an ore, but it also causes a change in its richness.

Lead veins appear to be more influenced by the nature of the country rock than almost any other kind of mineral veins. Thus in Yorkshire lead is contained in fissures running through limestones, shales and grits.

When the opposite walls are limestone, the lode is good.
" " " " " grit and limestone, not so good.
" " " " " shale and limestone the lode is worse.
" " " " " shale and shale the lode is worst.

In Cardigan and Monmouthshire where the rocks are hard and gritty the lodes are poor, but where they pass through slaty grey rock which is fissured and shistose they are good.

At Llanrwst however a lode was best in a hard grey gritty rock and at the new Llangynog lead mine it was best in a hard porphyritic rock.

In Spain there was more silver in the galena from slaty rocks than from the limestones.

At Lake Superior the copper lodes in the amygdaloid were 2 ft. wide and productive, while in the greenstone, conglomerate and sandstone they were thinner and relatively not so productive.

Sometimes lithological changes cause even greater alterations, for example a vein which was rich when in one class of rocks, may on crossing into a new kind of rock be so poor that it is worthless. In Cornwall when granite is soft it said to be "*kind*" and generally when granite is coarse grained and where it is not pure but contains accessory minerals it is richer than in other parts.

In Cornwall when a lode crosses an elvan it becomes either richer or poorer. At Wheal Alfred the lode was poor in slate but rich in elvan. The lodes are richer in light coloured slates containing mica and quartz. The hard brown slates yield tin, while the blue slates yield copper.

Striking examples of this are to be found in the silver veins of the *Fahlbands* of Norway. At Kongsberg in Norway the *Fahlbands* are N and S bands of schist (quartzose, with mica, hornblende, chlorite) containing iron pyrites, magnetite and blende, which run through the gneissic and slaty rocks of the district. When the E. W. veins which carry silver run through the Fahlbands, they are rich. Similar phenomena are observable where quartz veins carrying gold cross the bands of Berezite (a kind of granite) at Berezovsky in the Urals.

Not only may the character of a lode be changed when it passes from one kind of rock to another, but it may also be altered by passing through varieties of the same kind of rock.

For example, the lead veins of Cornwall are usually richest where the slates are soft and fissile, and poor where they are slaty and hard. Or again Von Cotta tells us, that where we have granites and gneiss, together with mica schist and grey gneiss, silver ores shew a preference for these latter rocks rather than for the former. If we reflect upon the fact that the conductivities of different rocks both for heat and for electricity will have been different, it is not difficult to understand that differences in the country rock should have produced differences in the minerals which have been deposited.

Influence of the walls of a lode upon its contents.— The influence which the walls of a lode have upon the contents of the lode, is evidently very closely connected with the more general influence produced by the country.

There are many instances where the mineral has been deposited more upon one wall of a lode than upon another as at Mount Catini (See Dip p. 5.) That the minerals upon the two walls are different is not uncommon. Thus at the Rammelsburg mine on the west wall a grey mixture of iron pyrites, copper pyrites, galena and blende was found while on the east wall there was a yellow mixture of iron and copper pyrites.

Slickensides.

The movements to which a lode has been subjected may be learnt from slickensides and faults. The former are represented by polished and striated surfaces of ore or gangue. These are generally only a few inches in area. Sometimes however, as at the iron mines of Nijni Tagil in the Urals, they may be seen upon faces of ore, the area of which must be reckoned in yards rather than in inches. By observing these carefully, the direction in which a movement has taken place may be determined. They are sometimes observed in coal mines near to faults.

The regularity or irregularity, together with the intensity of this movement, may also often be made out. As these movements have rendered the ground loose, and in some cases brittle, the mines in which many of these slickensides occur, are often dangerous for the workmen.

At the Gang mine, Cromford, Derbyshire, slickensides have split themselves off with violent explosions. At the Eyan copper mine they have exploded by scratching.

Faults. (slides, heaves.)

A more important class of movements which ought to be enquired into are those which have produced a discontinuity in the material of the lode. These are known as faults.

Some of these have been produced by fractures which have intersected a bed or lode, like a more or less inclined plane.

The portion of the lode and the strata which were superincumbent on the plane, are the parts which generally have suffered displacement. The movement will usually have been down the upper side of the plane. In exceptional cases the movement may have been the opposite and a "reversed fault" produced.

The direction of movement may be sometimes seen by striated surfaces, traces of mineral on the surfaces along which movement has taken place, by the direction in which strata have been bent &c.

By carefully studying and tabulating these movements rules for the guidance of the miners may be established. It must be remembered that these movements have generally taken place both

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horizontally and vertically, the two motions being usually combined. Schmidt who wrote on faults in 1810 gave a rule respecting the direction in which a miner ought to turn on meeting a fault. In 1828 Zimmerman who generalized on Schmidt's theory, gave a geometrical interpretation of the rule which may be stated as follows.

In order to find the continuation of a lode turn to the side on which a perpendicular to the fault or cross course falls with regard to the line of intersection of the lode and cross course drawn in plan—that portion of the line of intersection and the perpendicular being considered which lie at the side of the cross course opposite to the side on which the known portion of the lode lies.

Striation on the walls of a faulting fissure often shew that the movement has not being down the dip but obliquely across it, and farther as pointed out by Professor Höeffer there may sometimes have been a partial revolution of the mass on the hanging wall of the fault. Perpendiculars to such striations ought—unless the revolution has been accompanied by a movement of translation—to give the center of the motion.

Sometimes the fissures of these faults are filled with clayey material or flucan, at other times with gangue and ore, so that we have the appearance of one lode having intersected and dislocated another. In addition to earthy matter and ore, there may also be many empty spaces which become reservoirs for gas or water, rendering it a necessity where such faults exist, to approach them with caution.

Lastly we may have a dyke of igneous rock acting as a dislocator, which is of common occurrence in some of our coal fields. In some instances these dykes have not only destroyed the continuity of a seam of coal, but they have coked, charred, and reduced it to cinders, for considerable distances on each side of the line of intersection (see p. 27.)

The throw of a fault may only be a few inches or it may be several thousands of feet. In the latter case it is measured or its existence is proved by the relative positions of certain geological strata. The horizontal and vertical displacement will vary along the length of a fault, at one end of which it may be zero. The throw and width of a fault may vary with the nature of the rocks. In soft rocks faults are sometimes narrow and shew striated surfaces like slickensides.

Sometimes the strata are bent in the direction of the throw, and "dip to the downthrow" and "rise to the upthrow." A fault crossing a synclinal or anticlinal might throw the beds on one side of it relatively to corresponding beds upon the other side, either away from or towards each other.

Such an effect is observable when two lodes dipping in opposite directions are heaved by the same cross course, as at Wheal Alfred, where the horizontal distance apart of two lodes on one side of a fault is different from the horizontal distance apart of the same two lodes on the other side. In calculating the quantity of material in a lode or bed beneath a given area it must be remembered that an ordinary fault may diminish that quantity while a reversed fault might increase it,—it being assumed that the faults are not absolutely perpendicular. Faults like lodes may have branches or they may dislocate each other. A trough fault may be described as two faults hading or underlying towards each other. Reversed faults may be explained on the assumption that a wedge shaped portion of the earth's crust with its apex downwards has been caused to slide upwards by a horizontal thrust, while an ordinary fault may be explained by the sliding upwards of a wedge shaped piece with its apex upwards.

The following table gives the results of observations made in Cornwall by Mr. Henwood of 272 intersections.

Of all the lodes the proportion intersected but not heaved is.....	2.7 or 2.7 %
Of the tin lodes the proportion intersected but not heaved is.	18.0 „
Of lodes yielding both tin and copper „ „ „ „	37.2 „
Of copper lodes „ „ „ „	17.7 „

Of all the lodes the proportion heaved towards the right hand is.....	13.5 or 5.0 „
Of the tin lodes proportion heaved towards the right hand is.	56.0 „
Of the lodes yielding both copper and tin „ „ „ „	44.0 „
Of the copper lodes „ „ „ „	52.4 „

Of all the lodes the proportion heaved towards the left hand is.....	8.0 or 3.3 „
Of tin lodes the proportion heaved towards the left hand is.	26 0 „
Of lodes yielding both tin and copper „ „ „ „ „	18.6 „
Of copper lodes „ „ „ „ „	29.8 „

Of all the lodes the proportion heaved toward the greater angle is..... 18.1 or 6.7 %
Of the tin lodes the proportion heaved towards the greater angle is..... 52.0 „
Of lodes yielding both tin and copper the proportion heaved towards the greater angle is..... 56.0 „
Of copper lodes the proportion heaved towards the greater angle is..... 74.2 „

Of all the lodes the proportion heaved towards the smaller angle is..... 3.4 or 1.3 „
Of the tin lodes the proportion heaved towards the smaller angle is..... 30.0 „
Of lodes yielding both tin and copper the proportion heaved towards the smaller angle is..... 6.8 „
Of copper lodes the proportion heaved towards the smaller angle is..... 8.8 „

The mean distance of the heaves of all the lodes is..... 16.4 ft.
„ „ „ „ „ „ „ the tin lodes is..... 15.4 „
„ „ „ „ „ „ „ lodes yielding tin and copper..... 14.6 „
The mean distance of the heaves of copper lodes..... 17.5 „

The mean distance of the right hand heaves is..... 18.7 „
„ „ „ „ „ left hand..... 12.0 „
The mean distance of the heaves towards the greater angle. 19.3 „
„ „ „ „ „ „ „ „ smaller „ 17.1 „

The percentage of heaves in different rocks.

	In Granite.	In Slate.
The intersections without heaves.....	26.2 %	21.5 %
Heaves towards the right hand	52.4 „	50.5 „
„ „ „ left hand.....	21.4 „	26.0 „
„ „ „ greater angle	65.6 „	64.0 „
„ „ „ smaller angle.....	8.2 „	14.5 „

The mean distance of all the heaves is..... 16.4 ft.
„ „ „ „ „ „ „ in Granite..... 17.1 „
„ „ „ „ „ „ „ „ Slate..... 16.3 „

The 233 intersections consist of 125 veins of clay or flucan and 108 quartzose veins or cross courses. They are distributed as follows.

	In Granite.	In Slate.
Flucan.....	78.7 %	47.0
Cross course.....	21.3 ,,	53.0

From this it is seen that in granite the cross veins are generally flucans while in the slate they are quartzose.

From other tables Mr. Henwood shews that the extent of heaves is in direct proportion to the width of the lodes and to that of the cross vein.

It appears as pointed out by Mr. Henwood, that the general rule is that the heave of the same lode by the same cross course is in the same direction at all depths. Exceptions to this however occur, as at Gunnis Lake.

The heaves towards the greater angle chiefly occur with flucans.

The age of faults and lodes.—

If a fault or lode cuts through a series of rocks A but does not pass upwards into a set of rocks B, we may assume that the lode is younger than A but older than B. For reasons like this we may say that the E and W veins of Cornwall which break through the Trias are younger than that formation, while the N and S veins which break through Cretaceous rocks are younger than the Cretaceous. Many of the veins in California are younger than the Jurassic. When a vein intersects another vein it is evident that the intersected vein is the older of the two, or if a vein contains fragments of a bed or of another vein, the vein containing these fragments must be younger than the bed or vein from which they were derived.

As examples of veins of relatively recent age Phillips gives the following.

At Aveyron in France veins of argentiferous galena traverse Liassic strata. In Algeria similar lodes are of Cretaceous age. Auriferous quartz veins at Vöröspatak in Transylvania in Tertiary sandstones. Near Volcano in Amador county California, there is a quartz vein cutting beds of sand and gravel, while at Steamboat Springs in Nevada the writer has seen veins in the process of formation.

When a lode B passes through but not beyond the walls of a lode A and the continuation of B is somewhere to the right or left of the point of intersection, it would appear that as B was thrown by A, that

B was the older lode. It may however happen that A is older than B, the throw of B having been due to a movement along one wall of A after it had been intersected by B.

Backs of lodes, basset, broil, gossans &c.—If the material of which a lode is composed is harder than the rock through which it runs, by processes of degradation it may be left standing up like a ridge or wall. In the Atlas Mountains lodes of spathic iron with barytes stands up above Cretaceous rocks. Other examples are in the lead lodes of Cardigan and Montgomery, certain quartz lodes in Australia &c. On the other hand if it is softer, it is worn away to form a hollow. In both cases its direction will be more or less marked across the country through which it courses.

In the latter case it may form a channel along which water will percolate, in consequence of which we may have a line of luxuriant, or of some particular kind of herbage springing up.

In many cases the exterior portions of a lode will be more or less decomposed to form a mass of friable earthy matter, which from the presence of iron is generally of a red colour. This is called a *gossan.* (*eisennerhut. Gr., chapeau de fer. Fr., pacos, colorados, Sp.*)

These gossans ought to be carefully examined as they often give a good idea of the nature of the minerals which will be found at a lower depth. For example at the surface we might have earthy red oxide of iron, while lower we might find oxide of iron with oxide and carbonate of copper and native copper, still lower copper pyrites. Generally in gossans we find oxides and oxidized salts, combinations with water, carbonic acid, phosphoric acid, bromine, chlorine &c., but no sulphides. Copper pyrites may change as follows 2 (Cu Fe S.) $+ 11\ O = 2\ Cu\ SO_4 + Fe_2\ O_3$; or galena after becoming a sulphate may suffer farther transformation in the presence of calcium carbonate into lead carbonate and calcium sulphate. Native metals often occur in gossans thus at Lake Superior native copper and silver were chiefly found near the surface. In Cornwall the average thickness of the decomposed portion of a lode is from 30 to 40 fathoms, but there are instances where it has been as much as 150 fathoms in thickness. When a lode is very porous, and contains much iron pyrites we may expect to find a good gossan. In such cases it will be of a red colour. If the color is dark and quartz is absent the gossan is considered good

Sometimes the gossans are of a bluish green colour, due to the decomposition of a copper ore, or greenish and brownish from a lead ore. These colours however are rare, the one produced by iron generally masking all others. According to its structure and colour a gossan is described as *sugary*, *cindery*, *irony &c.*

By carefully observing the phenomena which gossans present to us, much may be learnt about the transformations to which mineral matter is subject.

If the back of a lode reaches to the surface, the soil which lies upon it will generally be more or less affected, and the growth of vegetable matter will be prevented. In consequence of this we shall see along the line of a lode either barren strips of soil, or a peculiar vegetation.

In most countries a good gossan is regarded as being indicative of a good lode, and there are cases on record where a mine has been taken up from the appearance of its gossan. In Cornwall it is said that,—

"A lode will ne'er cut rich and fat,
Unless it have an iron hat."

The depth to which a gossan may reach greatly depends on the drainage level.

Gaetzschmann gives a long list of plants indicative of saline spring. Dumps of red and brown hematite or clay iron stone grow a special vegetation chiefly malvaceous. The Calamine hills of Westphalia have a peculiar flora, a most noticeable plant being *Viola Calaminaria*. In Michigan, Wisconsin and Illinois we have the lead plant *Amorpha canescens*.

An iron ore vein near Siegen can be traced for nearly two miles by birch trees growing on its outcrop. In Montana an *Eriogonum* is said to be indicative of silver ores. An analysis of this plant shewed the presence of arsenic (see Indicative Plants by W. Raymond. Trans. Am. Inst. Min. Eng. Vol. XV. p. 644).

When certain minerals are decomposing a slight odour might be produced, this being especially noticeable when the weather is warm and showery.

Over the backs of certain lodes, lambent flames are said to have been sometimes visible, an appearance which may have been due to decomposition. If these decompositions give rise to heat, we

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might expect that snow would not remain so long over the backs of lodes as at other places. And it may also be remarked that as the contents of a lode may be better conductors of heat than the rocks of the neighbourhood, independently of the effects of decomposition, more heat will constantly be conducted out of the ground along the back of a lode than at other places.

These together with other appearences and phenomena which the backs of lodes present to us, are well worthy of our attention, as it is by them that we are guided when searching for mineral desposits.

Distinction between bedded veins and beds.—

It may sometimes happen that a fissure has been formed parallel to the stratification, which opening has subsequently been filled like a lode.

As deposits of this kind have often great resemblances to those members of a stratified series which are included under the head of beds or layers, it is necessary that the observer should be on the alert to seize those signs which may aid him in making the necessary distinction. Thus for instance if the deposit has a banded structure, we shall know that it is a vein with which we are dealing. Or again, if it should send off branches into the surrounding strata, or if it includes fragments of the superincumbent rocks, we should also see that it could not belong to a series which had been regularly stratified.

Filling of Veins.—It has already been pointed out that fissures which when filled with mineral matter constitute mineral veins, may either be one of the results of secular contraction or they may have been gradually opened and at the same time filled by the crystallization of mineral matter along a primary fissure.

The following theories refer to the formation and filling of veins.

1. **Contemporaneous Formation.** This theory implies that a lode was built up at the same time that the rock which encloses it was formed. This theory fails to explain any of the many observed phenomena which lodes present.
2. **Igneous Injection.** In this theory it is assumed that veins were formed and filled by the injection of igneous matter from below. In support of the view that certain veins may

have been filled in this manner we have the fact that many minerals may be produced artificially during the process of fusion. Ebelmen and Berthier by decomposition in the presence of fluxes like boracic acid produced crystals of corundum, emerald, quartz, etc. We also known that many metals occur in eruptive rocks. At Lake Superior there are copper lodes in amygdaloid, but as pointed out by Phillips, if the copper had been deposited from igneous fusion it would have been alloyed with the silver with which it is associated. Some of the lodes at Schemnitz may be of eruptive origin.

The mullock veins of Australia are decomposed eruptive dykes with gold and silver in their joints,—the concentration of metal having probably taken place in conjunction with the decomposition.

Notwithstanding the above facts it is only in an extremely few instances that the hypothesis of igneous injection is sufficient to explain the phenomena presented by lodes.

3. **Electric Currents.** It seems possible that metals may have been precipitated in lodes from solutions of their salts by means of electric currents. Mr. R. W. Fox experimented on the electric currents which he observed in the lodes in Cornwall. In a summary of his views he remarks that with the electrical decomposition of earthy and metallic salts the bases would be deposited at the electric negative pole but a rock which was electro negative in one position might be electro-positive in another.

The experiments made by M. Becquerel and independently by Mr. Fox have shewn how varions metals may be precipitated by feeble currents and how materials like clay, plaster of paris, and even soft sandstone may become laminated and generally alter their structure.

At Pennance mine the current flowing between a copper and a pyrites lode was found to be sufficient to produce electro-type copies of medals, electro magnets, and even to occasionally yield faint sparks. Many of these currents may be the result of chemical decomposition in a lode or of chemical action at the earth plate. The most elaborate investigations on this subject made during recent years are those

carried out by Carl Barus on the Comstock lode. The hypothesis was that as currents (probably hydro-electric) had been flowing for many years, such currents had become constant, equipotential surfaces might therefore be discovered and mapped. The result of the work indicated that in certain instances an electrometrical survey of this description might lead to the discovery of ore bodies (United States Geolog: Survey. Monograph III.)

Thomson has said that thermo-electric currents in the earth can never have been of any appreciable magnitude, but the electric currents in the earth which have caused terrestrial magnetism, or which may have been an effect of terrestrial magnestism, are quite competent to produce electro-metallurgical processes of the greatest magnitude.

Under particular circumstances by electric actions, metals might be successively precipitated from a solution containing a mixture of salts. The order in which these precipitations would take place must be sought for by the laws of electrometallurgy. Such an action would also give a possible explanation of the sequence in any bands we might observe.

If material is being deposited by electric actions we must remember that such currents although on the whole travelling in one direction more than in another, go oppositely every day. This therefore whilst tending to give a greater deposit on one side than on another, might possibly be employed to explain why we sometimes find the same metal deposited on two sides of a lode.

4. **Aqueous deposition from above.**

That fissures were filled by the flowing in of metalliferous solution from above was a theory propounded by Werner. It received great support from Mr. W. Wallace who wrote on the lead veins in the north of England. Amongst many other observations Mr. Wallace points out that the lead chiefly occurs in the rocks favourable to the descent of water, while at great depths which are unfavourable for the descent of water the veins contain but little lead.

5. **Sublimation.**

At Nagyag in Transylvania metallic arsenic has been deposited on the lower faces of crystals of diallogite. This is a fact supporting the

view that at least a portion of the material in such a vien has been deposited by sublimation. That metallic minerals like iron glance are formed on the lavas of certain volcanoes is also a fact supporting the same theory. Many minerals like magnetite, galena, blende have been formed in smelting furnaces. Durocher and other experimenters have succeded in forming many minerals by passing metallic vapours through heated tubes. Notwithstanding these facts it would seem that the cases where metals have been deposited by sublimation are extremely rare.

6. **Lateral secretion.**

The theory which best explains the phenomena of ordinary lodes is that their contents have been deposited from solution, the solutions having been mineralized by percolation through the country rock.

Among the numerous facts which support this hypothesis may be mentioned.—

1. The contents of many veins are similar to the rock through which they run. Thus in limestone we find veins of calcite, in silicious rocks veins of quartz &c.

2. All waters contain mineral matter in solution. In the waters from mines many of the elements have been detected. Copper as sulphate is sometimes found in sufficient quantity to be of commercial value. In sea water, silver, lead, copper, iron and zinc have been found. It is said that the Munz metal on ships which have been for some years at sea has in certain instances increased in the percentage of silver it contained. The author found that this was not the case with the sheathing of a vessel which had been for some years in the North Pacific.

   At Steamboat in Nevada, silica and metallic minerals like cinnabar may be seen in the process of deposition from hot water.

3. The artificial production of many minerals by deposition from solution as for instance in the experiments of Senarmont.

4. The fact that the country rock of metalliferous regions contains metallic elements. The proof that there is a relation between the mineralogical contents of mineral veins and the nature of the rocks is due to Prof Sandberger. Instead of

making a general analysis of the rock Prof: Sandberger separately analysed its mineral constituents and in this way he found nearly all the elements which occur in veins.

Thus in olivine,—iron, nickel, copper, and cobalt were found. In augite,—copper, cobalt, iron, nickel, lead, tin and zinc have been found. The micas yielded many metals. Copper is found in nearly all clay slates.

Where organic matter is present as in the Mansfield slates it has probably acted as a reducing agent, converting sulphates into sulphides, precipitating silver gold &c.

5. Many of the phenomena presented by veins as for instance their banded symmetry which is so often observed.

7. **Solution and ascension.**

In this theory it is assumed that the material which fills up a lode has been brought in solution from great depths and not from the rocks in the immediate vicinity of the lodes. The heat and pressure at great depths would certainly favour solution.

In looking at the various theories which have been advanced to explain the formation of mineral lodes, which have been enumerated in the order given by Phillips, the one most adequate to explain the various phenomena we meet with in lodes is that of lateral secretion. As to the exact manner in which matter has been precipitated in veins we have but little definite information,—in one instance it may be due to cooling, in another the precipitation may be chemical while in another it may have been electrical.

II. BEDS AND LAYERS.

Their formation and subsequent alteration.—

The beds and layers with which the miner has to deal, are those members of a stratified series which are made up of useful minerals.

At the time of their deposition they were more or less horizontal and uniform in character over large areas, the only variations to which they would be subject, being similar to those we might expect in a sediment which was being formed upon the bed of a lake or of an

ocean; for instance we might expect the material which was accumulating near a shore line to be coarser than that which was being deposited in deeper water, also there might be gradations in thickness &c. After a bed had been formed, by denudation and metamorphism it might suffer many changes.

Thus for instance channels of water running across a bed may wear it out into hollows. These may subsequently be filled with sand and extraneous material. By actions like these we get the appearances known to coal miners as *swells* and *knuckle saddles*, *trough saddles* or *channels*. Or again metamorphic actions may raise beds up from their original horizontal positions and bend them into gentle folds or into the most varied convolutions. When we observe that changes in position have taken place we generally find that chemical, lithological, and physical changes have also been induced.

By studying all these variations of position and of character which beds have suffered during and subsequent to their deposition, much of their history may be learnt, which is not only of value scientifically, but is also of value to the practical miner, enabling him to form rules both general and special for the district in which he is employed.

Beds of coal, ironstone &c.

The most common of bedded deposits with which a miner has to deal are those of coal. These occur in seams generally from 2 to 4 feet in thickness, but in some cases, as in the brown coal of Bohemia, as much as 80 to 90 feet in thickness.

In examining these beds their thickness, strike, dip, depth from the surface, with all their variations, ought to be observed. Also the partings, faults, and other irregularities, together with the various effects which they have produced upon the seam, must be noted.

Many kinds of iron ore are found in beds, the most common being perhaps the clay ironstone of the coal districts of England &c.

Beds and layers are also known as *strata*, *measures*, *sills*, *mines*, *bassets*, *delfs*, *girdles*.

The dip which may be measured by degrees or by so many inches per yard is sometimes spoken of as the *pitch* or *hade*. In connection with the dip we hear the terms, "strong dip" "rearing measures," when the dip is steep, while when it is gentle a seam is said to be "flat." The deepest portion of a set of workings are said to be on the "dip side," while the shallower workings are on the "rise side."

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As examples of Coal seams having a steep dip we may take those of Belgium, North France, Pennsylvania, North Japan &c. For examples of coal seams bent into zig zags the student may refer to the Anzin district. Irregular masses of coal are found at Creuzot, Montchain &c. (see Ponson).

Thickness of Coal.—In Bohemia, France and Hungary, there are beds of Lignite from 60 to 100 feet in thickness. The Bézenet seam is up to 200 ft. Brown coal at Zittau in Saxony reaches 154 ft. in thickness. In Staffordshire there is a bed of bituminous coal 10 yards in thickness. Seams which are ordinarily worked are from 4 to 10 feet in thickness. A seam 2½ ft. thick may be considered as thin. In Somersetshire seams 11 in. thick are worked. In the Hartz mountains a seam from 15 to 20 cm. in thickness is worked. Seams like these latter can only be worked when the coal is of a qualtity that will command a special market, as for the manufacture of gas or metallurgical purposes.

Irregularities and Impurities.—Most of the large seams as for instance the "ten yard seam" of Staffordshire are usually an association of seams, in which we find coals of varying quality which may be in juxtaposition or separated by bands of shale or sandstone or other worthless materials called "*partings*." In this manner certain seams may be subdivided into 20 or 30 distinct layers.

In a seam of coal at Pictou 116 inches thick, there are 14 layers of shale with a total thickness of 19 in. 21 layers of tolerably good coal and 4 layers of coal containing iron pyrities. From an example of this sort it may be inferred that in a given seam of coal we might find coal fit for making gas, for metallurgical purposes, suitable for domestic purposes, fit for coke, worthless coal, shale &c.

A heavy mass of impurity say, of sandstone, thinning away at its ends, occurring in a seam of coal is known as a *horse*, its two ends being respectively its head and tail.

Seams of coal are sometimes traversed by more or less pronounced systems of jointing which give it a tendency to split in definite directions. The planes of division are known as *cleats*, *faces*, *slynes*. If there are two sets of joints one may be called *backs* and the other *cutters*. In consequence of these fractures some coals break into cuboidal, rhomboidal or other regular forms.

If a seam of coal is in contact with an igneous rock it may be friable, earthy, cindery or like coke, the changes having been produced by the heat of the dyke. The altered coal is sometimes called *blind coal.*

It has been remarked that the lower beds in a series, or the lower portions of a given bed sometimes are more anthracitic in their character than the seams or portions lying at a higher level.

Again the beds which are the most tilted or which occupy the regions which have been most contorted are usually more anthracitic than those which are relatively flat and lie in the undisturbed regions. This is exemplified in the coal seams of South Wales and Pennsylvania.

Along its outcross a seam of coal may be so changed that it can hardly be recognized as coal.

Seams in a district. The number of seams in a given district or coal basin are very different. Thus at Newcastle there are about 40 different seams, at Mons in Belgium about 116.

Age of Coal. In Europe and America most of the coal which is worked is of carboniferous age but seams are known of various ages. Thus at Brora in Scotland and in Virginia there is a miocene lignite. In Japan most of the coal is tertiary or cretaceous.

Analysis of Coal. The following are maximum and minimum values for the carbon contained in various coals compiled from data given in Dana's System of Mineralogy. An analysis of Peat and Wood are added to the table.

		C	H	O	N	S	Ash.
Anthracite.	S. Wales	92.56	3.33	2.53	...	...	1.58
„	Pennsylvania	84.98	2.45	1.15	1.22	...	10.20
Caking coal	Pas de Calais	86.78	4.98	5.84	...	...	2.40
„ „	Zwickau	72.27	4.16	10.73	0.34	0.81	6.00
Non-caking coal	Valenciennes	90.54	3.66	2.70	...	...	3.10
„ „ „	S. Staffordshire	72.13	4.32	17.11		0.54	6.44
Cannel coal	Wigan	84.07	5.71	7.82	...	...	8.94
„ „	Tyneside	78.06	5.80	3.12	1.85	2.22	8.94
Brown coal	Hesse Cassel	71.71	4.85	21.67		...	1.77
„ „	Irkutsk	47.46	4.56	33.02	...	...	14.95
Peat		58.00	5.5	29.0	1.5		6.00
Wood		50.00	6.0	41.5	1.0		1.5

All ordinary coals contain from .5 to 1 or 2 per cent of water. The lignites may contain from 12 to 40 per cent of water. Anthracites have a S G of 1.3 or 1.4. Bituminous coals about 1.25 and lignites 1.2 or 1.1. From an examination of the above table it will be observed that during the process of the formation of anthracite from vegetable matter there is a loss of oxygen, hydrogen and nitrogen and therefore an increase in the percentage of carbon.

Formation of Coal. It is difficult to accurately trace the changes which vegetable matter has passed through to produce the different varieties of coal, which chiefly depend upon the purity of the coal, its degree of bituminization and the proportion of fixed and volatile matter it contains. Le Conte ascribes the origin of these varieties to decomposition of vegetable matter in contact with air and out of contact with air. Anthracite is an extreme form of coal produced by heat and metamorphic action.

Pengelly regards the formation of coal after the primary decomposition of vegetable matter, to have taken place in four ways.

By a dry process with heat (Ex. anthracite).

By a dry process without much heat (Ex. steam coal, Wigan cannel).

By a wet process with heat (Ex. bituminous house coals).

By a wet process without much heat (Ex. certain bituminous coals and lignite).

Iron Ore.

Clay iron stone occurs in beds as concretionary nodules associated with coal and coal shales in South Wales, and Staffordshire. Sometimes it occurs as a bituminous shale as for example the black band ore in Scotland. These ores are chiefly carbonates and contain 20 to 30 per cent of iron.

Beds of iron stone also occur in liassic and oolitic strata in Yorkshire, Northamptonshire, and Lincolnshire. These ores are hydrated oxides of iron containing 25 to 30 per cent of iron.

Hematite occurs in beds containing 60 per cent of iron as in Missouri, Michigan etc. These ores are usually siliceous. In Algeria there is a bed 5 yards thick which is said to contain 65 per cent of iron. At Cleator, Moor, Cumberland, there is a deposit 60 ft. thick.

Copper Ores. The Kupfer schiefer of Prussia is a stratified deposit of bituminous marly slate containing various ores of copper together with iron pyrites. galena, zinc, nickel, cobalt, silver, bismuth &c. At Mansfeld it is 11 to 20 in. thick and yields from 2½ to 3% of copper. Cupriferous shale beds have been worked in Shropshire and in China.

At Alderly Edge in Cheshire a bed near the base of the new red sandstone is saturated with blue and green carbonates of copper. It yielded 1½ to 3 per cent of copper.

At the Parys mine Anglesea, there are beds mineralized with copper. Copper is also precipitated from the water pumped from the mines.

Salt. In Cheshire 75 to 100 ft. thick. The Sperenberg bore hole passes through 3,890 ft. of saliferous strata.

Lead ore. The Bunter sandstone of Bleiberg yields nodular lead ore, chiefly galena but also cerussite.

Most of the beds containing minerals like copper and lead are the results of mineralization subsequently to the original formation of the beds.

Sulphur. Occurs in beds in Sicily, Greece, Yezo.—and other countries.

Surface deposits.

By surface deposits we mean those beds, of alluvium which more or less cover the face of all countries.

These have been chiefly created by various mechanical agents, which after having degraded the higher rocks carry the material which has thus been formed down to lower levels. By degradation most mineral deposits are so comminuted, that by their exposure to the atmosphere they are decomposed and destroyed. Substances like cassiterite, platinum, gold, &c. being exceptions to this,—that is not being so readily subject to decomposition,—have in consequence been more or less preserved and buried amongst these surperficial deposits.

In observing deposits of this kind, we must notice their general situation, area, thickness, and richness. Often we may have several beds ranged one above another, in which case we may have to deter-

mine their relative value. If we are tracing any particular deposit, as for example whilst ascending a valley, if the particles of ore increase in size and in number, we may expect that we are approaching their common origin. Another indication that we are near this point of origin will be that we shall find the mineral less worn.

In looking for a deposits of this kind, in a country where superficial deposits are known to occur, we may be often guided like the Tungusians in Northern Siberia, who search for gold by first looking at the general contour of the country, and observing those places where any obstacles like a projecting range of hills would be likely to prevent material being directly washed from higher to lower ground. Holes, sudden bends, or anything which would cause a diminution in the force of a current of water, are points at which we should expected that heavy material like gold or platinum would be likely to collect. In Australia although the most gold is generally found in *pot holes* and behind *hard bars*, it has often been found upon the shallow bends of ancient river courses. The lowest of a series of beds is generally the richest. This may be perhaps explained by a Hypothesis due to Mr. Belt who suggested that because large nuggets of gold are only found in the alluvial deposits, therefore the upper portions of the lodes or deposits which were first worn away, were richer than the lower portions. The more probable reason is that it is due to the action of gravity. Another reason for finding large nuggets in the alluvium may be due to their having been formed in situ by precipitation from a solution of chloride.

Chloride of gold is precipitated in the presence of vegetable matter with considerable rapidity. Gold has been found in fossil wood and in mine timbers.

By observing the nature of the associated minerals, which may be in the form of gravel or sand, we may often learn something about the rocks from which the deposit originated.

By carefully examining these surface deposits we shall find that like lodes, they have many peculiarities. Thus the alluvial deposits of gold in the Urals are said to be chiefly, and almost only, found on the Eastern flanks of these mountains,—a fact which points to the mountains of Asia, rather than to those of Europe, as being the source from which they originated.

In Australia the best *wash-dirt* is on the bed rock in the lowest parts of a *lead* called the *gutter*. The gravels of ancient river courses in which it occurs may be on the surface or at the depth or several hundred feet being covered by beds of basalt (blue stone) or even beneath a hill.

The largest nugget was the "Welcome stranger" 2,268 oz. In 1876 in Victoria 574,164 tons of drift averaged 22.67 grains of gold per ton; 35,938 tons of *cement* which required crushing yielded 4 dwt. 13¼ grains per ton. Certainly drifts have yielded 10 dwts. to 2½ oz per cubic yard.

In California the gold bearing beds are from six to several hundred of feet in thickness. They usually consist of gravels which may be cemented to form a conglomerate, sands, bands of tuff, clay, fossil wood etc. They from a band 60 miles wide along the western flanks of the Sierra Nevada. The big Blue lead extends 65 miles and has a depth of from 1 to 300 feet. Three or four cents per cubic yard will pay the expense of hydraulicing but near the base of certain leads as much as £ 10 per cubic yard has been obtained.

Magnetite occurs in alluvial deposits. In North Japan it seems to be uniformly distributed through nearly all the soil that is met with. In places it has collected together in beds and is smelted.

Bog iron and manganese ore which have accumulated by precipitation in marshy places or in lakes usually contain too much impurity to be of commercial importance.

Stream tin occurs in gravels much in the same way as gold.

B. IRREGULAR DEPOSITS.

I. SEGREGATIONS.

Segregations, like veins, occur in rocks of various descriptions. Sometimes they appear in a form which is more or less vertical approaching in character to a vein, at other times they approach in character to a bed.

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Under this head we have Stockworks or floors, may Contact deposits, and irregular deposits like those occurring in limestones.

Stockworks consist of a number of small veins of ore running in an irregular manner through a large mass of rock.

The relation of segregations to the rocks in which they exist is unfortunately not so definitely marked as in the case of veins, nevertheless certain relations may be sometimes observed, as for example at Tilt Cove in Newfoundland, where we have huge segregations of copper pyrites showing a preference to a particular band of serpentinous rock about 200 yards in width, whilst in the neighbouring rocks outside this band no metals are found.

Although the phenomena presented by segregations are difficult to understand, nevertheless they may perhaps be interpreted by a careful observation of the appearances which these deposits present us with.

II. IMPREGNATIONS,

The greater number of impregnations have probably been formed by the permeation of mineral matter through the rocks in which they occur,—the matter at the time of this permeation being in the state of gas, fusion, or solution.

It often happens that impregnations occur in the immediate vicinity of deposits like lodes, or segregations. In such a case it remains for us to decide whether the impregnation was derived from the mineral mass, or vice versa, not forgetting that it is also quite possible that the two might have been simultaneous formations.

From the manner in which the ores occur something may be learnt about the origin of an impregnation. Thus for instance some impregnations can only be regarded as the colouring matter of the rocks they penetrate, at once suggesting the idea that they originated from the infiltration of some solution. The red colour of certain sandstones which are coloured by a pelicle of oxide of iron surrounding each grain, are known to have been formed in this way. Again, the green coloured deposits of copper at Alderly Edge in Cheshire, may be taken as another example.

The impregnating ore may pass in films through a rock, or it may be distributed in grains and globules. Sometimes it may occur in crystals like the crystals of feldspar in a porphyry, a good example of

which we have in a deposit of galena on the Port au Port peninsula, Newfoundland, where the ore is distributed in small cubes through a mass of soft silurian limestone.

In some few instances we seem to have evidence of particles of mineral matter having been deposited together with sediment to form one of the members of a stratified series. Although the resultant beds might be said to be impregnated, it is perhaps better that they should be classed with the regular deposits.

Even if a bed should subsequently to its formation become impregnated with metallic matter, it may be convenient from its regularity to place it with the same group.

Examples of mineralized beds have already been given (p. 29).

As it is evident that the formation of an impregnation, must to a great extent be dependent on the physical and chemical nature of the rocks in which it has been precipitated, it is scarcely needful to remind the student that a study of those rocks may be fraught with the greatest interest, and valuable results may be obtained.

EXAMPLES OF IRREGULAR DEPOSITS

(*Segregations and Impregnations*).

1. Contact Deposits.

In Arizona red oxide of copper occurs between limestone and granite and slate rocks. At the Tanokuchi mine, Tosa, Japan, copper pyrites, between clay slate and diorite. Lake Superior, native copper between trap and sandstone.

At Chessy near Lyons, where there are ancient copper mines, deposits occur in and between metamorphic rocks and Bunter sandstone. In the metamorphic crystalline rocks there are yellow ores (copper and iron pyrites) grey and black ores (melaconite, iron pyrites etc.) which occur in a wedge shaped bed of grey rock between the crystalline rocks and the sandstone; red ores (cuprite) occur in clay against the sandstone; blue ores (azurite etc.) occur in nobules in the sandstone. The original deposit from which the others were derived was the yellow deposit.

Comstock Lode, Nevada.—Here there are silver ores between diorite and diabase. In places it is altogether in the diabase. Be-

tween 1859 and June 1880 it produced $175,000,000 of silver and $140,000,000 of gold.

Near Christiana at Narverud and Danemora near Upsala there are contact deposits of magnetite between granite and slate.

At Clausthal there are deposits of hematite between similar rocks.

Running N. and S. for 150 miles through Servia there is a band of syenite, greenstone &c. which traverses jurassic and cretaceous rocks. Between the eruptive rocks and the metamorphosed sedementary rocks there are many contact deposits yielding gold, copper and iron pyrites, galena, magnetite &c. They are described in detail by von Cotta.

2. Imbedded in Igneous or Non-stratified Rocks.

At Zinnwald (Bohemia) in a dome shaped mass of greisen (quartz and mica) there are horizontal narrow veins of tin, forming stockworks.

At Altenberg there is a greisen like rock called zwitter, throughout which tin is finely disseminated to the extent of ⅓ or ½ per cent. to form a stockwork. Somewhat similar deposits occurs in granite. In Cornwall small veins of tin were discovered at the china clay works near St. Austell. Some of the veins which run through granite consist of schorl and quartz. Near Stanthorpe in Queensland, grains of tin are disseminated through a granitic rock as if forming one of its chief constituents.

The carbonas at St. Ives are masses of coarse granite charged with tin and connected with the lode by a film of quartz.

At Monte Catini in Tuscany the cretaceous strata is broken through by serpentine and *gabbro rosso.* At the outcrop the lode was about 3ft. broad, but 12 to 30 feet below, it widened out and contained pockets with dimensions of 30 feet by 100 feet. The ores are erubescite, copper pyrites.

In Cuba and in Newfoundland, rich deposits of copper occur in serpentines.

In Cornwall, granites and slates containing 8 to 9 lbs. of tin stuff to the ton of rock yield a good profit. Sometimes 2 lbs. of tin stuff has been sufficient. A man can break down 1½ to 2 tons of hard rock per day and 7 to 8 tons of soft killas.

3. Imbedded in Stratified or Metamorphic Rocks.

At the Rammelsberg in the Hartz there is an ore bed which dips downwards and then forks like an inverted Y. The rock is clay slate. It has been described as a stockwork, a segregation, and as a bed. At one point its dimensions were 1,900 by 150 feet. Fifteen different metals are found, but the chief are galena, copper ore and zinc. It has been worked since A.D. 1,000.

In Andalusia, S. Spain, there are amygdaloidal masses of galena in metamorphic silurian limestones.

In the metamorphic rocks following the band of eruptive rocks of the Banat, there are many irregular deposits.

At Fahlun, Sweden, there is an enormous irregular deposit of copper pyrites (3 to 4% of Cu.) in metamorphic chloritic rocks.

4. Imbedded in Limestone.

The irregular deposits in limestone are chiefly those of lead, zinc and iron. In Derbyshire there are *pipe* and *flat veins* of galena following the bedding of the limestone, while small veins called *skrins* fill the joints. The deposits in Cumberland are very similar to those in Derbyshire.

In Flintshire at the Fawnog mine, galena and carbonate of lead form an irregular bed-like deposit between limestone and sandstone.

In Wisconsin, in cambro-silurian limestone, pockets and flat veins of lead occur.

The silver lead deposits in the dolomitic limestone of Leadville, were probably formed by the gradual displacement of the limestone and were not deposited in pre-existing cavities.

In Belgium (La Nouvelle Montagne near Verviers and near Aix la Chapelle) carbonate of zinc and galena occur in beds and pockets in limestone. In quartz slate, zinc carbonate and hematite are found.

In the Forest of Dean at the base of the Millstone grit and at the top of the carboniferous limestone, in fissures, in veins and in irregular deposits called *churns*, some of which measure 14 by 300 yards, hematite is found.

In Cumberland, in cavities in carboniferous limestone beneath millstone grits and boulder clay, large irregular deposits of earthy and crystallized hematite occurs, sometimes 60 feet thick (see p. 28).

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MINING OPERATIONS.

BORING.

Objects of Bore Holes.—Bore holes which are usually from 3 to 10 inches in diameter are sunk for the following purposes:

1. Testing the character of superficial deposits.
2. Exploring and testing mineral deposits.
3. Precautional purposes (10 or 15 feet in advance in soft ground; 2 or 3 feet in advance in hard ground).
4. Ventilation.
5. Obtaining water, brine and mineral oil.

One of the most complete treatises on Boring, is Tecklenburg's "Handbuch der Tiefbohrkunde."

Methods of Boring.—

Percussion boring.	With rods	Ordinary method of chipping and removing débris.
		Japanese method of plunging and not removing débris.
	With rope	Chinese and other methods with a spring pole.
		Ordinary method in American oil districts.
		Special methods of Mather and Platt and others
Rotary boring.	Boring with augers in soft materials.	
	Diamond rock drill.	
Hydraulic boring.	Applied to rotatory and percussion boring.	

Examples of Bore Holes.—

NAMES OF PLACES.	DATE.	DIAMETER.	DEPTH.	HEIGHT SPOUTED UP.	OBJECT.	QUANTITY PER DAY.
Sperenberg.		15in.-12in.	4194ft.		rock salt	
Mondorff.			2400		brine	
Grenelle.	1834-1841		1798		water	864000 gals:
Passy.	1855-1857	2ft. 4in.	1923	54ft.	,,	5582000 ,,
Kissingen.			1878½	48ft.	brine	

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NAMES OF PLACES.	DATE.	DIAMETER.	DEPTH.	HEIGHT SPOUTED UP.	OBJECT.	QUANTITY PER DAY.
Chicago.		5in.	1,700 / 1,000		water	800,000 gals.
Louisville.	1857 / 1858	3in.	2,086		,,	800,000 ,,
Charlestown.	1848		1,250	10ft.	water	16,700 ,,
Newsalwerk.	1858		2,288		,,	
La Chapelle.	1866- / 1869	5ft. / 5ft. 7in.	1,811		,,	
Creusot.			3,017		coal	
Place Hébert Paris.		3ft. 6in.	2,359		water	
Schladebach.		11in-1.22in.	5,732			

Operation of Boring.—Choice of place depends on geological considerations, character of the ground, presence or absence of water, nearness of roads, &c.

First sink a staple or pit 5 or 6 feet in diameter, if possible down to the solid rock. Dig a hole and place in it the guide tube (of wood or iron), one foot of which may project above the ground.

The amount of turn given to the boring rods between each blow is from one-eighth to one-sixth of a revolution. The length of the stroke will vary with the hardness of the rock and weight of rods;—in clay about 6 inches, in limestone about 24 inches. Ponson gives as limits, .15—.6 m. which are practically the same.

Rate of Progress.—In soft alluvium the first 30 or 50 feet may be easily bored in one day.

The number of strokes is generally about 18 or 20 per minute.

In soft rock a chisel may penetrate 6 inches without resharpening, but not more than 2 inches in very hard rocks.

A sludger will descend 1000 ft. in 3½ minutes. Pumping requires 4 or 5 mitutes and it may then be raised by a windlass in 12 or 15 minutes.

Rates of progress in different rocks in 11 hours:—(*André*).

			Progress.
1.	Tertiary and Cretaceous strata to a depth of	100 yards	1 ft. 8 in.
2.	Cretaceous strata, without flints. ,, ,, ,,	250 ,,	2 ,, 1 ,,
3.	,, ,, with flints, ,, ,, ,,	250 ,,	1 ,, 4 ,,
4.	New red sandstone ,, ,, ,,	250 ,,	1 ,, 10 ,,
5.	New red sandstone ,, ,, ,,	500 ,,	1 ,, 5 ,,
6.	Permian strata ,, ,, ,,	250 ,,	2 ,, 0 ,,
7.	Coal measures ,, ,, ,,	200 ,,	2 ,, 3 ,,
8.	Coal measures ,, ,, ,,	400 ,,	1 ,, 8 ,,
	General average	275 ,,	1 ,, 9 ,,

Pass rapidly through weak ground, which ought to be quickly lined. During the operation of boring in soft ground, clay dropped into the bore hole gives strength to the sides.

Diameter of Bore Holes.

DEPTH.	DIAM. OF HOLE.
300–403 ft.	6 in.
1,000–2,000 ft.	8 in.—12 in.
Artesian wells.	16 in.—2 ft. 6 in.

If the ground is difficult, the diameter should be greater than if the ground is easy. Up to 600 or 700 feet boring may be done by 6 or 7 men. For greater depths a small engine 6–16 H. P. will be required.

Rods.—Iron rods may be square, octagonal or circular in section. Their length is usually 12–15 feet, some times 25 feet. With screw joints, the female socket is deeper than the male plug. In passing rods down a bore hole the socket in the lower end goes downward. Collars ought not to touch. Iron rods one inch square in section weigh 10 lbs. per yard in air, but 8.7 lbs. per yard in water. S.G. of wrought iron = 7.7.

For a 3½–4in. hole, 1in. square rods may be used to a depth of 90–100 fathoms with rotatory boring; 1in. square rods to a depth of 50 fathoms with percussive boring; ¾–⅞in. square rods to 100 fathoms, if a free falling cutter is used.

1 inch square rods may be 12 feet in length.

1¼ " " " " " 15 " " "
1½ " " " " " 18 " " "

Lengthening rods are multiples of the lengthening screw.

1 inch square rods have screws 1¼″ in diameter, 2½″ long.—7 threads to an inch.

Wooden rods are made of straight pine 20–35 ft. long with a square section and not less than 2½ inches in the side.

At Schöningen (2013 feet), Rods 2⅞″ diam., 40′ 6″ long, 3½ lbs. per foot in weight. Lowest 228½ ft. of rods were of wood and weighed 902 lbs. in air (786 lbs. in water.)

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The remaining 1,784½ ft. of rods weighed 7,007 lbs. in air (2,047 lbs. in water.)

The latter when of iron weighed 8,676 lbs. in air (7,042 lbs. in water.)

When of wood the weight to be raised was 2,833 lbs. or 1.35 lbs. per foot and was raised in 1½ hour. When of iron, they weighed 7828 lbs. or 3.9 lbs. per foot, and were raised in 3–4 hours, so that the saving in time by using wood was 56 per cent.

To obviate the severity of shock to a tool in a deep bore hole, hollow rods may be used.

Sliding Joints and Free Falling Cutters.—In 1834 Œynhausen invented the sliding joint. Kind's free-falling apparatus acts by the resistance of the water in the bore hole. The action of Degousée's cutter depends upon a portion of the apparatus resting at the bottom of the hole. Fabian's free falling cutter depends upon a twist being given to the rods. Dru's contrivance acts by a sudden shock being given to the rods.

Chisels or Bits.—The form, the sharpness and the temper of the edges of cutting tools are modified according to the rock to be cut. The various forms of chisels are known as flat or straight chisels for ordinary rock, the diamond pointed or *V* drill for hard rocks, the *T* chisel for gravel. × + *Z* and *S* chisels (cross and club borers) are difficult to sharpen. They may be used in inclined fissile strata. For earthy materials augers are employed. An annular arrangement of cutters is used for core boring.

Sludgers are usually from 3 to 10 feet in length. The valves are flat or balls. These may be weighted to suit the consistency of the sludge which is to be entered. The rope to raise the sludger is ¼ to 1 in. in diameter.

Accessory Tools.—Verifying tools, clearing tools, regulating tools, guides to prevent vibration, stirrup with screw or chain for rotating and gradually lengthening the rods, a tiller or brace head, lifting dogs, tigers, nipping forks, dogs &c. A parachute may be employed to prevent a sudden fall of the rods.

Extracting Tools.—For small articles a spider box, Kind's catch clutch or a ball of clay kneaded with oil and hemp placed in

the end of a sludger or screw bell may be sufficient; otherwise they may be slowly chipped to pieces or dissolved by acid. Broken rods may be extracted by a crow's foot, screw bells, box bells, &c.

Rocking Lever and Trestle.—The length of the lever may be 10—20 feet. Its leverage which is variable, is generally about 1 : 4 or 1 : 5. The lever may be worked by a crank or cam with steam power. The lever may be supplied with a balance weight and if a sliding joint is used an elastic stop will be required. To work the rods, a spring pole or a windlass and rope may also be employed.

Bore Frames and Towers.—These vary from 30 to 80 feet in height. Windlasses provided with brakes are required. If the hole is to be deep let there be two pulleys to work the rods. Diameter of the windlass about 20 inches.

Guide Tubes.—These are of iron or wood with shutters. In Japan the guide tube is attached to the bore tower above the hole. In Europe it is placed in the ground in the upper part of the hole.

Tubing.—Square and made of narrow planks; cylindrical of wood, or iron (cast or wrought) with screw, socket, lap or muff joints. If in corrosive water, of brass or copper. Iron tubes which have been tinned may be used for water supply. The lower end of the tube is sharpened. Tubes are put down by percussion or pressure. Tubes may be passed through running sand by forcing water through a series of holes in the end of a pipe passed down the inside of the lining which latter is under pressure. Tools may be used to enlarge a hole below a set of tubes. To raise a set of tubes, use a screw plug, Kind's plug or Alberti's tube drawer. Tools may be employed to cut tubes and thus draw them up in sections. Mandrels are required to straighten tubes. Clay may be used as a temporary lining.

A Set of Boring Tools for 500 Feet.

1 Pair tillers with spare screws.
50 Ten feet lengths 1¼ in. boring rods.
1 Short Swivel rod.
1 Gin wheel and frame.
1 Snatch block.
1 Spring hook and 30 ft. rope.
1 Auger board. 1 Auger cleaner.
Iron work for shear legs. 1 Windlass.
1 Clay Auger, each 4¼, 5¼, 6¼ in.
1 Flat Chisel, each 4¼, 5¼, 6¼ in.
1 V " " " " " "
1 T " " " " " "
1 S " " " " " "
1 Shoe nose shell, each 4, 5, 6 "
1 Auger.
1 Worm Auger. 1 Crows Foot.
1 Bell Box. 1 Spiral Worm.
2 Lifting Dogs. 2 Hand Dogs.

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Difficulties and Accidents.—Rods become brittle by blows and consequent vibration. Rods, chisels, bolts, &c., by carelessness may fall into the hole. By want of rotation projections may be left and the hole looses its circular form. Deviation of the hole from the vertical. Falling in of the sides of the hole. Jamming of the apparatus. By change of strata or fissures, the hole may become narrower. Breaking and falling in of sludging apparatus.

Japanese Method of Boring.—This is chiefly applicable to alluvial deposits, which from the surface are penetrated at the rate of 50 or 60 feet per day. The tools consist of solid round iron rods connected by fish joints and terminating with a pear shaped plunger for soft ground and an obtuse pyramidal point for hard ground. By a rocking lever terminating with a *Jizai kagi* or *kettle catch* (peculiar to Japan) the rods are pumped up 6 inches at a time to a height of from 2 to 15 feet when they are allowed to fall. Material is but seldom taken out of the hole. During the operation the hole is kept filled with water charged with clay. Subsequently it may be lined with a bamboo pipe. The method is cheap and rapid.

Rope Boring (American System).

The disadvantages according to Löttner are :—

1. The impossibility of employing ordinary chisels.
2. The uncertainty in the rise and fall of the tool.
3. The necessity of having rigid rods in case of hindrance and breakage.
4. The impossibility of rotating the tool in soft clayey ground.
5. The small effect compared with that obtained by the free falling apparatus.
6. The impossibility of knowing the effect of the borer by holding the rope.

Notwithstanding the above objections, from the writer's own observations, the American system of Rope Boring is probably one of the most practical in existence. In Pennsylvania several thousand wells each from 1,000 to 1,500 feet in depth are bored through shales and limestones every year. At starting, a hole is 8 in. in diameter but

when lined is about 5⅝ in. The boring bar is 70 ft. in length with a sliding joint near the middle. A small engine works the walking beam at about 60 strokes per minute, subsequently it raises the tool and works the sludger. Two men carry on the operation. The total cost of a 1,500 ft. hole as given by Prof. A. Lupton is as follows :—

	Dollars.
Carpenters' Rig (Derrick or pulley frame with housing for borers, boiler, &c.	350
Belt, bull rope, engine telegraph, water and steam pipes, and boiler fittings	100
Boiler 20 h.p. Engine 15 h.p. on ground	750
Contract for drilling Contractor to furnish fuel, tools, cable, sand pump, line, &c., at 65 cents a foot for 15,000 ft.	975
Casing, say 300 ft. at 80 cents	240
Tubing, say 1,600 ft. at 20 cents	320
Torpedo (almost invariably used before tubing)	100
Packer	25
Working barrel	8
Casing head	3
Tees and elbows	5
One 25 barrel tank	25
One 250 barrel tank	110
Tank house	25
Expenses of tubing and packing well	20
Expenses of hauling material	50
Total	£650 = $3,106
Extra for pumping tackle	100

Apart from plant the cost of a 1,500 ft. hole is only $750 or £200, and this where wages are from 10*s.* to 16*s.* per day.

Mather and Platt's Rope-boring Apparatus.

The chief features of this apparatus are :—

1. The contrivance for rotating the rod.
2. The arrangement for giving the blow.
3. The details of the sludging apparatus.

Many holes have been put down in various rocks 300 to 1,200 feet deep at average rates of 1′ 6″ to 4′ per day.

Diamond Boring.—The average rate in stratified rocks appears to be from 5 ft. to 6 ft. per day, and this for holes from 1,000 to 1,300 feet deep and 18 inches in diameter.

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In diamond boring a number of black diamonds are fixed on the end of a steel cylinder attached to tubular rods. By suitable machinery on the surface the rods are rotated, water being pumped down the rods to wash up the debris. The chief advantages are rapidity and obtaining solid cores of the rocks penetrated.

Diamond Rock Drills are largely used for prospecting in many mining districts as for example at Cletor moor in Cumberland (Hæmatite), Australia (Gold), Japan (Coal), Pennsylvania (Oil), Michigan (copper).

The diamond drill will cut through the hardest rock. The following table is given by André as the ordinary rate of progress :—

Granite	and	hardest limestone	2 to 3 inches per minute.
Quartz	„	„ „	1 inch per minute.
Sandstone	„	„ „	4 inches per minute.

Difficulties are met with in rocks which are irregular in their character like conglomerates,

Boring Journal (*André*).

Boring executed at —— in the County of ——
Began —— 189—. On the Estate of ——
Completed —— 189—.

Date.	Description of strata.	No. of specimen in case.	Thickness.	Depth from surface.	Angle of dip.	Diameter of bore hole.	Description of tool employed.	Time actually occupied in passing through.	Quantity of water met with.	Organic remains.	Remarks.

Cost of Boring.

The cost of the diamond boring, exclusive of the cost of lining :—

8*s*. a foot for the first 100 feet.
16*s*. „ „ „ „ second 100 feet.
24*s*. „ „ „ „ third 100 feet.

so on in arithmetical progression, increasing 6*s*. at each 100 feet. Above 1,500 feet in depth, special rates are charged.

The formula for finding the cost of a diamond bore hole:—

$$s = .5\,d\,(0.4 + 0.004\,d)$$

s = sum in pounds sought.

d = depth in feet.

André gives the following formula to calculate the cost of ordinary boring, in pounds sterling.

3*s*. 6 *d*.	a yard for the	first	10 yards.
7*s*. 0 *d*.	„	„	„	„	second	„	„
10*s*. 6 *d*.	„	„	„	„	third	„	„

From the above it is seen that the increase is in arithmetical progression.

Generally, if,

s = sum sought in pounds sterling.

d = depth of the boring in yards.

$$s = 0.5\,d.\,(0.175 + 0.0.175\,d).$$

These examples of the cost of boring are only given to illustrate the method of calculation. From the writers own experiences with bore holes in Japan they cost rather more than they do in Pennsylvania, notwithstanding the fact that wages in Japan are only about 1/15 of what they are in the United States (See p. 42).

To the above, the cost of sinking the staple must be added. When extra hard rocks are met with, the cost will rise at an extra rate per fathom, or by day's wages. It may be stipulated that the employer may abandon a boring if conditions arise necessitating a change in the contract. Further it may be arranged for the boring master to receive an extra sum for every week that the boring has been completed before the stipulated time, which sum may be that paid for the last week of work or a certain proportion of that sum. If the stipulated time is exceeded, a reduction say of 25 per cent. may be made on the sum due to the boring master. For neglect or unnecessary delay the employer may break the contract.

Cost of Boring Tools (*Bower Scott and Western*).

In the following table the depths are given in feet, and the price for tools in pounds sterling. Windlass, with one each of different dimentions of Augers, Shells and Chisels are included. Shear legs not included:—

Depth.....	30	50	100	200	300	500
Price.....	20	35	60	85	130	230

Compare these prices with prices on p. 42.

Precautions.—Keep the bole hole plumb, otherwise its deviation from the vertical may increase.

Keep the bore hole circular by regular rotation. Keep a careful record of all that is mentioned in the headings of the boring journal.

Examine the rods for flaws. Keep them straight and let them be numbered. Test the temper of tools. See that their edges are symmetrical.

From time to time obtain specimens, either as *chips* or *cores* from the bore hole.

Do not allow a tool to lie at the bottom of a hole, but during the time when work is not going on raise it a few feet.

In attempting to withdraw rods which have been jammed, use steady pressure and light blows. Sudden shocks may result in fracture.

If a bed is reached at a depth much greater or much less than was expected, it must be remembered that the difference may be due either to one large fault, to many smaller ones, or to a change in dip.

Precautionary Bore Holes.—There are holes bored horizontally in advance of levels approaching old wastes which may be filled with water. One hole is kept in advance of the level, while others may be driven obliquely right and left. The length of these holes which are 1¼ to 1½″ diameter depends on the depth from the surface and the hardness the rocks:—

Depth from surface in fms..... 5 10 15 20 25 30 50 75 100
Length of hole in advance in yds..... 6 7 8 9 10 11 15 20 25 (*Collins*).

To determine the force which may be applied to the Tillers of Boring Rods (*Humber*).

The moment of resistance M to torsion of a rod or bar is given in the following formulæ:—

For a solid circular section $M = \frac{fd_1^3}{5.1}$

For a hollow circular section $M = \frac{f(d_1^4 - d_2^4)}{5.1 d_1}$

For a solid squares section $M = .28 f a^3$

Where d_1 and d_2 are the external and internal diameters of the rods, a is the side of the square, in inches and f is the resistance of the material to shearing in pounds per square inch of section. The values for f are as follows:—

Wrought Iron	50,000
Cast iron (average)	27,000
Oak	2,300
Fir	500 to 1,500
Ash and elm	1,400

The force acting on the rods, is that which is applied by the men at the end of the tillers.

It may be expressed $M = Pl$

where l = length of lever in inches

and P = force applied to lever in lbs.

This force ought not to exceed one quarter of that calculated by the above formulæ.

The first of the above formulæ finds more practical application in determining the diameter of shafts, drums, pulleys, &c.

Thickness of a Bed.—Having obtained the dip of a bed $\theta°$ and its apparent thickness a, its true thickness may be calculated as follows.

True thickness = a. cos. θ.

Dip of bed.—A bed has been reached at three points from a horizontal plane at depths of x, y and z feet respectively; to find its dip.

1st Solution. Plot the points x, y and z in plan; join them. Draw the plane containing these lines. The inclination of this plane taken at right angles to its horizontal trace is the dip required.

2nd Solution. If I be the true dip of a bed, i the apparent dip in a direction $\theta°$ away from the direction of true dip then,—

$$\text{Cos } \theta = \frac{\tan i}{\tan I}$$

Therefore if we plot $x\,y$ equal to the tangent of its inclination to the horizontal, and $x\,z$ equal to the tangent of its inclination to the horizontal, each in its proper direction, and from y and z erect perpendiculars to meet in o, then $x\,o$ shews the direction of the true dip and measures its tangent.

3rd Solution (*Merivale*). Let $A\ B\ C$ be three bore holes.

S = angle of dip of bed.
V = angle between the strike of the bed and $A\ B$.
a = distance $A\ B$
a' = distance $A\ C$
W = angle in a horizontal plain between $A\ B$ and $A\ C$.
d = difference in depth of A and B
d' = „ „ „ „ A „ C

$$\text{Tan } S = \frac{d'}{a \sin V}$$

$$\text{Tan } V = \frac{\frac{da'}{d'} \sin W}{a - \frac{da'}{d'} \cos W}$$

The Quantity of Water.—The quantity of water flowing from a bore hole made in an old waste may be calculated as follows:—

Let H = the head of water in feet.
d = the diameter of the bore hole in inches.
l = length of the bore hole in feet.
G = number of gallons delivered per hour

Then.

$$G = \sqrt{\frac{(15d)^5 H}{l}} \text{ (Hawksley).}$$

It must be remembered that the head decreases, and therefore to determine the time taken to empty a given reservoir it is better to calculate the times taken to empty so many successive horizontal sections of the reservoir. A more practical method is to dam up the drift and allow the water to escape through a notch of known dimensions (*See Drainage*).

Survey of Bore Holes.—The most satisfactory instrument to determine the angle of deviation of a bore hole from the perpendicular and the direction of deviation is Macgregor's clinograph. This consists of a small pendulum of glass floating by means of a hollow glass float inside a bulb, and a magnetic needle similarly floating in a lower bulb. The fluid in the bulbs is a hot solution of gelatine These bulbs are fitted in a protecting tube. While the solution is hot they are lowered in the bore and there allowed to cool. On withdrawal from the hole they can be placed at the angle at which they cooled and the inclination of the bore and its azimuth of deviation determined.

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BREAKING GROUND.

VARIETIES OF GROUND.

1. Rolling, running or watery (quicksands).
2. Mild, soft, easy (clay, chalk coal).
3. Jointy, scaley (vein stuff and fissured rocks).
4. Fast, hard (like the above but without joints). (In these rocks explosives are required).
5. Unusually hard materials like quartz, iron pyrites.
6. Soluble materials (rock salt).

Weights and Bulk of Material.—The weight of a cubic foot of any material in lbs. is its specific gravity × 62.425 or the weight of a cubic foot of water. One ton of quartz when solid occupies about 13 cubic feet, but when broken about 20 cubic feet. Rocks when solid as compared to the same when broken usually increase in volume in the ratio of 1 : 1.5 or 1 : 1.8—the increase being dependent on the size and form of the fragments. A knowledge of these facts may be of use in calculating the dimensions of a tub to carry a given weight of coal, in calculating the quantity of cement that will be absorbed in the interstices of broken rock when making a given volume of concrete, &c.

With fine materials there may be a decrease in bulk by the addition of water. Sand may in this way loose 20 per cent of its volume, therefore a cubic foot of wet sand ought to weigh more than a cubic foot of dry sand. The writer however finds that this is only true when the sand is weighed in the box in which it was moistened. If wet sand is loaded into a vessel it packs with difficulty and therefore in practice it may be found that a load of wet sand is lighter than a load of dry sand.

MINING TOOLS

(*See Manual of Mining Tools by W. Morgans*).

Shovels, Spades.—Shovels are flat or dished, with a central ridge or crease and straps. For loose materials they are large and

square, otherwise they are pointed. Width of plate or blade from 8 to 16 inches. It is made of steel, or iron with a steel edge and it meets the helve which is long (4 or 5 feet) or short (30 inches) at an angle of 140° to 160°.

Scrapers are used with a box, tray or basket.

Picks. Pikes. Mattocks. Mandrels. Slitters or **Hacks.**—The head of wrought iron with steel tips. Cast steel heads are liable to fracture.

A pick in its action, is a combination of a hammer, wedge and lever. Points are square or chisel edges.

The heads may be straight, sweeped, elbowed or anchored.

For overhand work, square work and holing, light straight picks are used. For underhand surface work, heavy picks with a little sweep are employed. Weight of heads from 2 to 8 lbs. Helves 24 to 33 inches.

Poll Picks used by metal miners, have one stem of 12 inches and a stump of 3 inches to form the poll. Weight 4 lbs. To avoid wincing the eye should be long and oval at ends. Rectangular and round eyes are sometimes used. Eyes are punched across the laminæ of the bar. To resist wincing, corbel bits or straps may be employed.

Patent picks have movable tips or heads. They are only serviceable for light work

Varieties of Picks.—Picks for holing, benching kirving or undergoing. Bottom picks. Striping mandrels or picks for pulling down. Driving, stone picks or twibills &c.

A smith and boy sharpen 70 to 100 tips of collier's picks per hour, and perhaps 40 to 60 tips of metal miners picks.

Hammers and Sledges.—Large hammers used with two hands are called sledges.

For single handed boring, hammers of 2¼ to 4 lbs. are used. For double handed boring the sledges weigh 4 to 10 lbs. usually 5 to 7 &c.

The striking face or pane which is slightly convex, is usually of steel.

Varieties.—Bully pattern, block, bloat, plug, dolly which is circular or many sided. Lump sledges, cobbing hammers, bucking irons, spalling hammers.

A smith and two strikers can forge eight 7 lb. sledges per day. A smith and striker can resteel 12 sledges per day.

Cast steel sledges are said to give a smarter blow.

Borers, Drills, Augers.—A Jumper is a bar of iron with a steel bit at each extremity and a heavy bead in the center, dividing the tool into two stocks. It has a lively action and is good for vertical holes.

Steel borers are more effective than those of iron, a short borer or an old borer is more effective than one which is long or new.

The edges are straight or curved. They should not be "nipped," "oddcornered" or partially "backward."

For good borers it is necessary to have,—good steel, good shape, good tempering—when good fuel is required, good striking and turning.

Diameter of hole or width of bits in inches.	1	1⅛	1¼	1½	1¾	2	2¼	2½
Diameter of stock in inches.	⅝	¾	⅞	1	1⅛	1¼	1⅜	1½

Angle of cutting edge from 60° to 100°.

Borers are sometimes sharpened with a swage. The colour for tempering varies from straw to purple, the former for hard ground and the latter for soft.

A smith and striker sharpen 45 single borers per bour. Sometimes borers wear away at the striking end more rapidly than at the bit end.

Varieties. Swallow tail bit, club bit, nicker bit, S bits, X bits, Z bits. These latter are difficult to sharpen.

Revolving bits and augers are used in benching down coal, also in earth and other soft materials.

Gads which are pointed and wedges with chisel edges are from 3in to 2ft in length.

Helves, Handles, Shafts, Sticks.—Helves require to be light and elastic.

Ash and hickory make good helves.

For picks and sledges, the cross section is oval. Round cross section for shovels.

Helves are split out of logs, being afterwards hewn and dressed, so that the long axis of the feather radiates from the log.

Sawn helves are apt to break. The average life of a good collier's helve is 4 months. In metal mines a helve may only last 2 weeks.

BLASTING.

Miscellaneous Tools used in Blasting.—Scrapers to clear boring dust, meal or sludge out of a bore hole. Drag twist, loop drag, swabstick or sludgers, for cleaning a hole. Scoops and spoons for charging powder. Clay iron, shooting needle or nail, pricker. Tamping bar which is tipped with copper or made of bronze. Shooting needles and prickers ought also to be made of bronze. Tamping case.

Tamping or Stemming.—Above the charge place a small quantity of some soft material, after which fill up with loose rubbish which ought not to be quartzoze.

Hard clay, broken brick, or shale are good materials for tamping.

Dry sand, plaster of paris, a plug of wood so cut that the effect of an explosion is to tighten it in its position, have been employed. Humble has invented an india rubber tamping but it is costly. Johnson's plug is of phosphor-bronze.

A tamping made of coal dust may if blown out into a dangerous atmosphere result in an explosion. Blown out shots in certain coal mines are always a source of danger—the flame extending when gas is present as much as 15 yards.

Fuses.—A rush, reed, paper or quill filled with powder and terminated with a piece of touch paper may be employed. The ordinary material is Bickford's patent safety fuse. This is made of tape or yarn, with a specialy prepared slow burning powder as a core. Guttapercha and metallic covered fuse is employed for subaqueous explosions.

Some varieties of fuse like white fuse, give but little or no smoke and flame, other varieties are impervious to damp. (Red fuse, sump fuse, tape fuse). Specially covered fuses are required in hot countries. A patent Quarry fuse conveys the fire to the *bottom* of a charge.

By using these fuses a better effect is obtained, there being no hole left in the tamping, time is saved, the danger of a pricker is avoided and there is little smoke or smell.

Electrical Fuses.—Quantity fuses explode by ignition of a material like meal powder round a platinum wire which is heated. The platinum wire, which has a high resistance and a low specific heat, is heated by the passage of a current from a battery. The leading wires which may be of copper do not require perfect insulation.

Tension fuses explode by the ignition of a chemical compound (chlorate of potash, subsulphide and subphosphide of copper) caused by the passage of a spark. The spark may be produced by a magneto-electric machine as Wheatstone's, Breguet's, Saxton's, Clarke's, an electro-dynamic machine like Siemen's, Ladd's, Farmer's, Gramme's, a frictional machine-like Bornhardt's, or a secondary current from an induction coil. The former machines are more convenient than the latter. Perfect insulation is needed on the line. The resistance of the line has little effect on the current.

With quantity fuses, the line must have a low resistance or many fuses can not be exploded simultaneously. The state of the line may at any time be tested by a feeble current. Small dry batteries may be used.

With electric fuses several charges may be ignited simultaneously. The same result may be obtained by a patent fuse which flashes from the point where it is ignited down to where it enters the charge. Several of the *flashing fuses* are united, and at the point of junction they are simultaneously fired with a piece of ordinary fuse.

Firing Explosive Substances.—By the application of heat dynamite simply burns, but by the explosion of a small quantity of substance like fulminate of mercury in the dynamite, the latter explodes with violence.

Detonators are copper caps containing fulminate of mercury. They are placed on the end of a fuse. By the explosion of the

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detonator the explosion of the substance with which it is in contact is accelerated. Different substances require different quantities or kinds of material to produce detonation. With gunpowder detonation may be a mechanical action, while with chemical compounds like nitroglycerine it may be the result of molecular vibration.

For complete detonation a charge ought to be confined. If loose, only part of it may explode.

The following table by Messrs. Roux and Sarrau shews the advantage of detonation.

Substance.	Simple Explosion.	Detonation.	Relative Weight of Gases.	Heat disengaged by 1lb.	
				Simple Explosion.	Detonation.
				units.	units.
Fulminate of Mercury.	.....	9.28	.....	.....	1354
Gunpowder	1.00	4.34	0.414	1316	1318
Nitroglycerine.....	4.80	10.13	.800	3097	3200
Guncotton	3.00	6.46	.850	1902	1909
Picric Acid	2.04	5.50	.892	1491	1563
Picrate of Potash	1.84	5.30	.740	1417	1534

Two or more charges fired simultaneously give a better effect than if they had been exploded separately.

For this reason other things being equal it is better to remove large masses of material by simultaneously firing a large number of small charges or even by one large charge.

In 1885 Hell Gate or Flood rock was removed by the simultaneous explosion of 14,000 cartridges of dynamite and a mixture of nitroglycerine and coaltar oil (rackarock).

The holes were 9 ft. by 2½ in. The material removed was about 275,000 cubic yards.

In this case 42,331 lbs. of dynamite acted as a detonator for 240,399 bs. of rackarock.

Relative strength of Explosives (*Noble*).

	Weight for weight by mortar test.	Bulk for bulk computed from the 1st column.
Gunpowder	1.00	1.00
Guncotton	2.84	2.57
Dynamite	2.89	4.23
Nitroglycerine	4.00	5.71

EXPLOSIVES.

A rending compound is one where the gas resulting from explosion is developed gradually, as in gunpowder. Here explosion is occasioned by combustion.

In a shattering compound the gas is generated rapidly, as in dynamite. In this case explosion is due to detonation.

Cundhill gives the following classification of explosives:—

1. Gunpowder ordinarily so called.
2. Nitrate mixtures other than gunpowder.
3. Chlorate mixtures.
4. Nitro-compounds containing nitro-glycerine. Dynamites.
5. Nitro-compounds not containing nitro-glycerine. Guncotton.
6. Explosives with picric acid or picrates. These are strictly nitro-compounds.
7. Explosives of the Sprengel type, in which the principle is the admixture of an oxidising with a combustible agent (at the time of or just before being required for use) the constituents of the mixture being themselves non-explosive.
8. Miscellaneous explosives.

Gunpowder is a mixture of about 75 % nitre, 15 % carbon and 10 % sulphur. Sometimes nitrate of soda is used. This may be recognized by its rhombohedral crystals. The explosive force of gunpowder may be reduced by mixing it with sawdust, chaff, filings,

lycopodium, burnt lime &c. Glass powder which may be removed by sifting, renders powder inexplosive. A decrease in the quantity of nitre or an increase in the size of grains makes a slow burning powder suitable for blasting. The sulphur accelerates the combustion. When fired the resulting gases are, nitrogen, carbonic oxide, carbon dioxide, potassium sulphide. The latter is the white smoke.

All the constituents for powder are specially prepared. They are ground and sifted separately. Next mixed, made into a paste, pressed, granulated and finally sorted. Blasting powder is glazed with graphite which retards ignition. Compressed powder is sometimes used.

The strength of powder may be tested by the distance to which it projects a ball from a mortar. By the velocity of a projected ball, as observed either directly or as calculated from the deflection of a ballistic pendulum. By tests with a differential Mercury gauge, ordinary blasting powder gives 22,000 lbs. and sporting grades 40,000 lbs. per sq. in. Temperature causing explosion 600° F.

Good powder ought to be regular, hard, clean (not soil the hand or paper) and dry. A small quantity ought to explode on paper without soiling the same.

Powder containing nitrate of soda readily becomes damp. Good powder gives greater space for tamping than bad powder.

Schultze's Powder.—Consists of nitre and grains of wood-pulp converted into nitro-cellulose. It is said not to be injured by moisture, to be stronger than ordinary powder and that the resultant gases are not injurious.

Haloxyline is made from wood or vegetable fibre and nitre. It is hygroscopic and not so effective as ordinary powder.

Neumeyer's Powder is a slow burning powder with less sulphur and more charcoal than ordinary gunpowder. It may be useful for coal and salt.

Küp's Powder is like ordinary powder but with a large proportion of barium nitrate. It burns slowly.

White Blasting Powder consists of chlorate and prussiate of potash and cane sugar. There are several varieties.

Nitro-glycerine.—Discovered by Sobrero in 1847. Brought into use by Nobel.

It is liable to explosion by concussion, especially when frozen, (freezes at 4° C.) or in consequence of spontaneous decomposition.

Its use is forbidden in England, Belgium, Sweden and Austria.

Nitro-glycerine is formed by the action of nitric acid on glycerine. It is a white colourless liquid with a sweet aromatic taste. S. G. 1.S and therefore sinks in water. Three varieties, (mono,-di,-and tri-nitroglycerine) may be produced, the last being the one employed for blasting. Insoluble in water but soluble in alcohol. Solidifies at 45° F=(5°C) and decomposes at 420° F. With a higher temperature it explodes. At some mines it is prepared on the spot a short time before being used. Its solution in alcohol is inexplosive. The alcohol may be removed by washing.

If frozen it must be thawed by placing in a vessel of hot water. Its contact with the body is injurious. The gases produced by an explosion have a bad effect on the health of miners.

When exploded, 1 vol. of nitroglycerine gives 1,300 vols. of gas, and if this is expanded 10 times by the heat, we get 13,000 vols. of gas. The production of this is very sudden.

It requires a small bore hole and its effect is great, no tamping required and the presence of water does not prevent its action.

Dynamite consists of nitroglycerine mixed with kieselguhr, a silicious infusorial earth. Coal ash may be used instead of kieselguhr.

The Rhenish Dynamite Company have the following three varieties :—

	Percentage of Nitro-glycerine.	*Kieselguhr.*	
IA	75—77	23—25	For the hardest rocks and small levels.
I	70—71	29—30	Hard rocks and large levels.
II	60—61	39—40	Soft rocks and assisting No. IA—removing side cuts &c.

When ignited it burns quietly unless it is in a confined space. It is exploded by detonation. A thin layer on stone or iron may be exploded by a sharp blow. When in bulk concussion produces no effect. At 46° F it freezes and can not be easily exploded. It therefore requires to be thawed (in a watertight can placed in a vessel of hot water). It is not hurt by water.

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Its use gives economy in labour, in boring and also in tamping,—loose sand or water being sufficient. Can be used under water or in watery ground. Does not give smoke. In using Dynamite, use small holes, increase the depth and angle. After a miss fire do not remove a charge unless water tamping has been used.

In tunnel work the rate of advance will be 3 or 4 times the rate obtained with gunpowder. It is safe to use. The gases developed are not injurious.

Price No. 1 Dynamite 2/- per lb.
Price No. 2 Dynamite 1/4 per lb. } In quantity 5–10 per cent discount.

Dynamite may be employed in submarine work, in removing trees, breaking down salamanders, in bore holes to increase the flow of oil or water.

Its strength as compared with that of other explosives may be determined by exploding a small quantity of the substances to be compared (2 to 7 grammes) in cylindrical holes (say 0.08^{m} deep by 0.01^{m} diameter) bored in blocks of lead. The volumes of the cavities produced in the lead may be taken as proportional to the strength of the explosives.

Fair estimates of the relative strength of dynamites and other explosives have been obtained from the diagrams given by seismographs, the charges being fired under similar conditions.

Other tests are with a Mortar and ball or with a pressure gaugue where the result of an explosion is to compress a cone of lead.

Brain's Blasting Powder.—Nitro-glycerine, chlorate and nitrate of potash, charcoal, coal, sugar.

Carbonite.—Nitro-glycerine, wood meal, potassium nitrate, organic sulphur compound. Used for fiery mines.

Geligníte.—Nitro-glycerine, nitro-cotton, wood meal, potassium nitrate. In appearance like Blasting Gelatine.

Fulmatin.—85 per cent nitro-glycerine and 15 per cent of a chemical mixture in place of kieselguhr.

Gelatine.—86.4 per cent. nitro-glycerine, 9.6 per cent. soluble gun-cotton, camphor 4 per cent. potassium nitrate. Properties very

similar to dynamite but more powerful. One variety is not unlike Forcite. It needs strong detonators.

Lithofracteur contains nitro-glycerine, kieselguhr, powdered coal and other substances. A black plastic material. Properties similar to Dynamite.

Dualine.—Nitro-glycerine mixed with sawdust which has been treated chemically. Many of its properties like dynamite. When confined it may however be exploded by simple ignition.

Giant Powder.—Nitro-glycerine, sodium nitrate, resin, sulphur, infusorial earth. Also a name used for Dynamite.

Judson Powder.—Nitro-glycerine, sodium nitrate, resin, surphur, asphalt, coal. Also a form of Dynamite.

Atlas Powder.—Nitro-glycerine, sodium nitrate, wood fibre, magnesium carbonate.

Hercules Powder.—Nitro-glycerine, potassium nitrate, and chlorate, magnesium carbonate and white sugar. A form of Dynamite.

Ætna Powder.—Nitro-glycerine, &c. A form of Dynamite.

Different grades of the above are manufactured which vary in the percentage of nitro-glycerine.

Michalowski's Blasting Power.—This is a chlorate compound with manganese binoxide and organic matters.

Picric Acid Compounds.—Oil of tar treated with nitric acid yields crystals of picric acid the salts of which are explosive. The powders containing these salts have not been much used. Abel's powder, Designolle's powder, Melinite.

Gun-cotton.—Gun-cotton is prepared by treating cotton or cellulose with a mixture of sulphuric and nitric acid. After this it is well washed and treated with a dilute alkali.

It may be exploded by a smart blow and it ignites at temperatures between 50° and 100° C.

It is prepared as solid or hollow cartridges, granulated, as cord and as paper. When not confined it only burns. The gases after an explosion are CO, CO_2, CH_4, N, H_2O, and H. The fumes of CO render the use of guncotton in mines objectionable.

Preparations of guncotton soaked with nitrates and chlorates have been prepared.

Tonite is a mixture of barium nitrate and gun cotton.

Roburite is a mixture of ammonium nitrate and chloro-nitro-benzine. Used in the Wigan coal district.

Securite is nearly the same at Roburite and Bellite.

Bellite is a mixture of ammonium nitrate and di-nitrobenzole.

Ammonite.—Here there is an excess of ammonium nitrate.

Amongst the smokeless gunpowders, we find:—

Amide Powder.—A mixture of potassium and ammonium nitrates and charcoal.

Nobel's Powder.—Nitro-glycerine, nitro cellulose, camphor.

Poudre Vieille.—Used with the Libel rifle.

Gaens Powder.—Nitro-cellulose, potassium nitrate, ammonium ulmate. Walthamite, Cordite &c.

For a complete list of explosives see "*A Dictionary of Explosives*" by Major J. P. Cundill.

Lime Cartridges.—Generate a steam pressure of 2,850 lbs. and expand to five times their original bulk. They are extremely safe and give neither smoke or smell. Sebastian Smith and Moore.

Water Cartridge's.—When an explosion is surrounded by water it may secure safety against blown out shots in a dangerous atmosphere. Settle and Abel's.

Position of Bore Holes.—If there is only one unsupported face, a hole requires to be angled. Thus in sinking a shaft or in driving a level, one or more holes angled in the center of the face to be removed, will form a *cut*, *key* or *core*. Successive holes may be driven at right angles to the face and arranged either concentrically round the *key*, or else parallel to the sides of the level. Rupturing holes are sometimes employed.

In driving a level where the dip is downward and towards you, work from above downwards, but when the dip is away from you, work

from below upwards. If driving parallel to the strike, work from an advanced cut, either to the right or left or across a level.

Line of least resistance.—The quantity of explosive employed in different bore holes in the same class of rock ought to be proportional to the cube of the line of least resistance.

This rule apparently implies that the weight of rock which is lifted is proportional to the resistance to be overcome. So far as rupturing the rock is concerned, the quantities of explosive employed in any two cases ought to vary as the square of the line of least resistance.

With a quick explosive where inertia is a factor in the resistances to be overcome, the quantities of explosive may be proportional to the length of the line of least resistance.

To increase the resistance along the line of tamping, a strong explosive occupying little bulk must be employed. The length of tamping may be increased by enlarging the bottom of a bore hole. In limestone rocks at Marseilles this has been accomplished by the use of nitric acid. The length of the line of least resistance should not exceed half the depth of the hole.

For gunpowder, one rule is to use half the cube of the line of least resistance measured in feet, as equal to the number of ounces to be employed.

The rule given by Molesworth is, $P=\frac{X L^3}{8}$

When L=Line of least resistance in feet.

X=Number of ounces of powder required when $L=2$ ft.

P=Quantity of powder required in ounces.

For large blasts see Hydraulic Mining p. 86, also p. 56.

Selection of an Explosive.—In selecting an explosive consider its strength, its safety, the effect produced upon it by moisture, heat &c., the gases evolved by explosion, and its price.

An explosive whose detonation temperature is less than 4000° F fired in a properly stemmed hole will not ignite fire damp. Under very favourable circumstances a temperature of 1200° F may ignite fire

damp. For safety the temperature of detonation should not exceed 2700° F. Nitrate of Ammonia detonates with a temperature of 1850° F and it is used with dynamite, gun cotton, &c. to reduce the temperature of detonation. The French Government has ordered the use of this class of explosives for fiery and dusty mines. H.M. Inspectors of Explosives adversely report on the use of ammonium salts except the carbonate, the reason being that ammonium salts by exposure become acid and nitro-compounds like gun-cotton, nitro-glycerine are affected by acids, and decomposition followed by ignition or explosion may occur.

The strength depends upon, the quantity of gases developed, the nature of the gases, the heat generated to cause expansion, and the rapidity of explosion. In similar bore holes the space occupied by equal weights of different explosives will be inversely as their densities.

	Specific Gravity.	Temp. of Explosion.
Gunpowder	1 or over.	2231° C.
Dynamite	1.5	2940° „
Cotton powder	1.5—2	———
Gun-cotton	1 or under.	2636° „
Nitro-glycerine	1.6	3170°

Contents of One Inch of a Bore Hole (*André*).

Diam : hole inches.	1	1¼	1½	1¾	2	2¼	2½	2¾	3
Gunpowder or Gun cotton. oz.....	.419	.654	.04	1.283	1.675	2.120	2.618	3.166	3.769
Dynamite. oz.....	.670	1.046	1.507	2.053	2.680	3.392	4.189	5.066	6.030

Force developed by explosion of Gunpowder (*André*).

When gunpowder is exploded 56 % of solid matter is formed and 43 % of gas, or the solid matter is to the gaseous matter as 6 : 4.

The gas produced at a temp. of 32° F is 280 times the bulk of the powder employed, therefore the pressure developed is $\frac{280}{4} \times 15 = 10{,}500$ lbs. $= 4.68$ tons. But the temperature is raised to

4,000° F (absolute temp.): or starting from an absolute zero (461.2° F) the temperature has been raised 8·11 times.

The pressure in the cavity is therefore 4·68×8·11 or 42·6 tons per square inch. Such pressures have been indicated by gauges.

COAL CUTTERS.

When we consider the unfavourable circumstances under which a collier works whilst hewing coal, we should naturally suppose that the work ought to be better accomplished by means of some machine.

One of the machines devised for this purpose was invented by Mr. Pierce, of Wigan. It consisted of a pick attached to a bell crank lever which was worked backwards and forward, by having one of its arms attached to a reciprocating piston rod of a small engine. The method in which it cut the coal was similar to the method in which it was cut by hand labour. For this reason the miners christened it the "Iron man." Like many other inventions, which are the first of a series, it did not realize its originator's expectations, and quickly fell into disuse. Since the death of the "Iron man" a number of other machines have been devised, and several of these are to be seen in daily use at several of our collieries. These machines have not however yet reached that perfection, which there is every reason to believe they are destined to attain, and whether any particular coal cutter works satisfactorily still depends greatly upon local circumstances.

From the following notes on Coal Cutters, which are largely taken from André's Coal Mining, it will be seen that these machines may be divided into three groups,—those working with a percussive chipping action like a pick, those which pare or cut the coal like a plane, and those which cut or scrape it away like a circular saw.

Commencing with the oldest type like the "Iron man" we have the following.

Firth's Coal Cutter.—This consists of a horizotal steam cylinder 5in. diameter 12in. stroke. The piston rod of this, works the arm of a bell crank lever, to the opposite end of which a pick is fitted. To allow of a certain regulation for the position of the line of undercutting, this pick can be fixed at various elevations in a slot. On the

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piston rod there is a tappet which gives a reciprocating motion to the slide valve. From this it will be seen that the pick must make its full stroke before it can be drawn back. The machine is advanced by hand, by means of gearing connected with the hind wheels of the frame or car on which the machine is carried. When working, the first cut is made with a pick of about 75 lbs. weight, working at the rate of 70 blows per minnte, to a depth of about 2 feet. After this, the pick of 75 lbs. weight, is replaced by a longer pick weighing 90 lbs., and the same cut is gone over again and is completed to a depth of about 3 ft. 5in. In 24 hours a cut of this depth has been made along a face 257 ft. in length.

In an improved form of the machine the whole depth is cut at one operation and the advance of the machine is automatic. The pick will cut in any position, and the back stroke of the pick, by making the front side of the piston of less area than the back, is more gentle.

The vibration to which the machine is subjected and the loss of time during the back swing of the pick seems to have rendered machines of this discription not so desirable as those which have been constructed upon other principles.

Schram and Oliver's patent coal cutting machine. This machine is percussive in its action, working a system of five or more picks making up to 400 strokes per minute.

Carret and Marshall's Coal Cuiter.—This consists of a small hydraulic cylinder 5in. in diameter, the piston of which makes a stroke of 18in. To the end of this piston there is a bar about 4 ft. in length in which 3 cutters are fixed. These cutters project one beyond the other like a series of steps. As this bar reciprocates it cuts its way into the coal. At the same time that the forward portion of this movement takes place, that is while the coal is being cut and when there is consequently a heavy back thrust upon the carriage carrying the machine, water is admitted into a second cylinder standing vertically, which in consequence drives its piston, the end of which is provided with a bearing surface against the roof. By this means the carrying carriage is clamped down tightly upon its rails. During the return stroke of the cutter, the clamping piston is drawn down and the carriage is automatically advanced ready for the second cut.

In this way about 20 cuts are made per minute, and during an hour, the coal will be undercut to the depth of 4 ft. 6 in. for a length

of about 20 yards. The water is under a pressure of about 300 lbs. By means of a hand wheel the cutting cylinder can be turned so that the bar may work into the coal in various directions, or whilst being removed from one point to another, may be placed parallel with its carrying car.

Economic Coal Cutter.—The essential portion of this machine is a long bar on which a deep screw has been cut. On the thread of this screw a number of cutting teeth are so fixed that when the bar is revolved against a face of coal it tends to cut its way in a direction at right angles to its length. By means of suitable gearing, this cutting bar can be canted or turned round. The advance is automatic.

The Legg Mining Machine.—This also works with a cutter bar with 26 bits on it, which is revolved.

Baird's Coal Cutter.—In this machine we have a small air cylinder 8½in. diameter with a 12in. stroke which by means of suitable gearing drives two horizontally placed star wheels. These carry an endless chain bolted into the links of which, are the cutters. In one hour it will undercut to the depth 3 ft. for a distance of about 15 yards. The motion of the machine along the face is automatic.

Winstanley and Barker's Coal Cutter.—This machine undercuts the coal in a manner similar to the way in which the work would be done by using a horizontally revolving circular saw.

In this machine 2 oscillating cylinders drive a vertical shaft, carrying a star wheel gearing into the teeth of a large wheel placed horizontally. This latter wheel, which is 3 ft. 6in. in diameter, has each of its teeth furnished with a socket, into which a cutter is bolted. The teeth of the star wheel are so large that they do not touch the cutters. The cutting wheel is carried on a large arm which can be turned out or in by means of a hand wheel turning a worm gearing in a rack fixed on to the arm at the opposite end from the cutting wheel. By this contrivance it will be observed that the cutting wheel may be caused to hole its way into the coal.

It will cut 3 ft. deep for a distance of from 20 to 25 yards per hour.

Gillot and Copley's Coal Cutter.—The general principle of this machine is similar to the one just described. One chief difference

is that the cutting wheel can not be pivoted in and out, being carried upon a fixed arm. This cutting wheel is driven by beveled gearing, in connection with 2 air cylinders. These which are 7in. diameter and 12in. stroke are placed side by side.

This machine makes a cut about 3ft. 4in. deep for a length of about 20 yards per hour.

General remarks on Coal Cutters.—All the machines which have been referred to with the exception of Carret and Marshall's coal cutter, which is worked by water power, are driven by compressed air. The reason that water power is required in the Carret and Marshall Machine is because the principle on which the cutters work make it necessary to apply a strong steady pressure, and as only small cylinders are admissible in underground machines, such a pressure could not well be obtained by means of air. As there is comparatively little expansion to be obtained when working with air, the engines which are used, are specially constructed. The ports are made large, the stroke short and the speed quick.

In nearly all these machines, quick speeds are reduced and power is gained, so that the cutting portion of the machine has comparatively a slow motion. This intermediate gearing although advantageous in giving a steady powerful motion, is not to be recommended on account of its liability to derangement. When we consider the circumstances under which a coal cutting machine works, it is evident that because it is so liable to derangement it ought to be made light and rigid, and at the same time its parts as few as possible. The advance of these machines is generally automatic, the machine hauling or winding itself upon a chain, one end of which is anchored at the end of the face along which it is cutting. The rails on which the machines run ought to be of the same gauge as those which are ordinarily used in the mine. In many cases the gauge is broader. The machine ought also to hold itself upon its rails or else be capable of being easily clamped. In Baird's machine it will be observed that the action of cutting, tends to drag the carriage towards the face which is being cut, while in Carret and Marshall's machine the carriage is being pushed in an opposite direction. Much may be done to counteract these forces by fixing the hauling chain in a proper position. Generally speaking it will be better the nearer it is to the face.

When circular cutters are used it will be observed that it will be necessary to make the floor as even as possible. The feed ought to be self regulating, and the cutter ought to be able to cut in various directions. Most of the coal cutters can only be used for holing or undercutting, and the slitting has to be accomplished by hand. As many parts as possible of a machine ought to be protected by a cover.

Bar cutters can be more easily made to undercut to a greater depth than disc cutters; they are not so likely to be jammed when working in tender coal, and as the bar can be replaced, new cutters are put in more quickly than they can be with a disc machine. Many disc machines do not cut their way into the coal. Bar cutters may be more conveniently driven by electric motors which run at about 700 revolutions per minute, the cutter making 500 revolutions. Amongst electrically driven coal cutters we have those of Edison, Goolden &c.

The advantages which we should expect to derive by using a well adapted coal cutter are many. The work would be accomplished quickly, and fewer men would be employed. As a consequence of a reduction in the working staff fewer lives would be exposed to accident. Most important of all, however, would be the great saving in coal due to the more perfect undercutting. When undercutting is accomplished by hand, say to a depth of 3 feet, a more or less wedge shaped opening is made, which at the front is 10 to 15in in height. The coal which in this way is reduced to a state of powder may in a thin seam amount to 25 per cent of the whole quantity which is excavated. Not only does a machine avoid a great portion of this waste by making a cut which is not more than 3 to 4in. in height, but by undercutting to a greater depth, it renders the process of wedging more easy, and a larger quantity of coal is extracted at one operation than can be extracted when the operation has been performed by men. Also we may remark, that a machine will almost equally well perform the operation of undercutting in a hard parting, as in the softer coal. For these and other reasons it seems certain that the time is not far distant when coal cutters will be more universally used, and it will only be in exceptional cases where the coal miner will be called upon to carry out the laborious and dangerous operation of undercutting.

Coal Wedging Machines.—The wedge of Bidder and Jones is a modification of the plug and feathers system used in Quarrying blocks of stone—the wedges corresponding to the feather being forced

apart by a hydraulic press. In Cox's wedge, the portion corresponding to the plug is driven in by a sledge. Another wedge is that of Grafton Jones. In the Sampson Coal Getter the wedges are forced in by a screw. With Burnett's Roller Mining Wedge, a wedge is drawn between rollers which force out feathers.

Stanley's Coal Drifting Machine.—This machine cuts an annular groove round the face of a drift leaving a core which is then broken down. At Nuneaton Colliery it is said to work well, the rate of advance being four times that of hand labour. Diameter of level 5′.4″.

Other tunneling machines are Brunton's and Beaumont's. The latter was used in the Channel Tunnel.

ROCK DRILLS.

The following notes on Rock Drills and air compessors are chiefly epitomized from more complete descriptions give by André in his "Mining Machinery."

Rock drills should be simple, light, strong, small; they should strike the rock directly, the stroke should be capable of alteration, sudden removal of the resistance should not cause injury, the rotation should be automatic, the feed should be regulated by the advance of the piston; they should be able to work with pressure, easily taken to pieces; finally they should be adjustable at any angle, and portable. Hand feeding is usual.

It is better to have a heavy blow produced by mass than by a high velocity. With a small mass and high velocity, vibration and wearing take place.

About 1845 M. Maus who was engaged in the construction of the Turin-Genoa line of railway designed a set of drills which were pressed backwards by means of cams against a set of springs; on being relieved, they flew forward and struck a blow.

In 1851 Cave introduced the principle of making a drill the prolongation of a piston rod worked by air or steam.

In 1855 Thomas Bartlett introduced a drill, consisting of a simple acting engine, the piston rod of which passed into a cylinder where at each stroke it compressed the included air. By the expansion of this air the drill which was attached to the piston rod was forced forwards. One of the first practical machines was introduced by Sommeiller in driving the Mount Cenis tunnel, his great improvement being to make the "feed" automatic.

The general nature of the working parts of a rock drill may be understood from the three following brief descriptions.

Ingersoll Rock Drill.—The tool is a continuation of a piston rod working in a cylinder. The back part of the piston is hollow and runs backwards and forwards upon a spirally grooved bar inside the cylinder. At the end of this bar there is a ratchet with a detent which will allow it to be turned only in one direction. As the piston comes back it twists itself upon the bar which is prevented from turning by the ratchet and detent, whilst during the forward motion because the bar turns with it, it goes out straight. This gives the requisite rotation to the tool.

By means of tappets and spindles the requisite motion is given to the valve in the valve box. As the tool cuts into the rock the fore part of the piston continually approaches a tappet. On striking it, the motion of the tappet partially turns a rod and this by palls and ratchets turns the feed screw and the tool is automatically advanced to its work.

Dubois-Francois Rock Drill.—This drill is one which is much used in Belgium and France. Its mode of action is based upon the machine of Sommeiller. The piston like the pistons of other machines has a larger area at the back than in front. This works in a cylinder into which compressed air is admitted by an ordinary slide valve. This valve is connected with two small pistons. When air is admitted into the valve chest, because the right hand piston is larger than the other, the valve moves to the right and air enters the cylinder. This drives the main piston and the boring tool forwards. During the time that this action is going on, air is escaping by a small passage to the back of the larger of the small pistons, and this continues until there is equilibrium on both sides of this piston. The air is now able to act upon the smallest piston and the valve moves to the left, and air enters the main cylinder at its front end, and the main

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piston makes its back stroke. During this back stroke an annular projection strikes a lever, which opens a valve and allows the air to escape from the back of the larger of the small pistons. The equilibrium of this piston being destroyed, it is again driven to the right and the action repeated.

In connection with the main air passages, are two small single acting pistons which have an alternating up and down motion. This is communicated through a rod to a ratchet wheel, and the requisite feed obtained.

Darlington Rock Drill.—The Darlington Rock drill, is a drill which deserves especial attention on account of its simplicity. All valve gear is dispensed with, and on account of the fewness of its parts, it is a machine which is likely to take a leading place amongst Rock Drills.

In the main cylinder the piston and its rod by working backwards and forwards upon a spirally grooved bar in connection with a ratchet wheel, receives the requisite twisting motion, as in the Ingersoll Drill.

Air entering at an inlet port, drives the piston to the right. As it goes back, it first covers the exhaust port and then opens an equilibrium valve passage. The air then acts on the back of the piston, and because this back area is larger than the front area, the piston before it reaches the end of its back stroke is driven forwards. When it passes the exhaust port, the air escapes, and the equilibrium passage is closed and the air again only acts upon the front face of the piston.

With a pressure of 40 lbs., 1,000 blows per minute can be given.

Amongst the many other drills which have been invented we have the Victor, the Royal, the Eclipse, the Barrow, the Cornish, the Excelsior, the Cranston etc. The following will serve as an example of the cost of rock drills. It is quoted from the price list of the Sandycroft Foundry near Chester.

The Sandycroft Patent Rock Drill.—This Drill is being used in Mines in various parts of the world with excellent results. Some of its principal advantages are durability, simplicity in operation, economy in the consumption of compressed air, and adjustability. The pressure necessary to drive the Drill may vary from 25 to 70 lbs. per square inch, and the number of strokes of the Drill, from 50 to 1,000 blows per minute.

Prices of Air-Compressing Engines and Rock Drills:—

Number of Drills Air-Compressor will supply.	1 3in. Drills.	2 3in. Drills.	4 3in. Drills.	6 3in. Drills.	9 3in. Drills.	12 3in. Drills.	2 4in. Drills.	4 4in. Drills.	6 4in. Drills.	8 4in. Drills.
Rock Drills ...	£45	£90	£180	£270	£405	£540	£120	£240	£360	£480
Air-Compressing Engine.	£130	£195	£380	£462	£635	£720	£380	£635	£820	£945

Electrically driven Rock Drills.—In one class of these drills the reciprocating motion is obtained by sending a current first through one solenoid and then through another, the iron core being attracted first in one direction and then in another. Owing to the heating of the solenoids this type of drill has not yet become successful.

In a second type, a motor by gearing, drives a cam which forces the drill back against a spring, which when the cam has passed, drives the drill forwards and strikes a blow.

Hand power drills.—In hand power drills by the revolution of a wheel turned by hand, cams are caused to force a piston behind the drill, back into a cylinder and then compress the included air. After the passage of the cam past its tappet, the expansion of the included air drives the drill forward. Amongst these drills we have Jordans and Burtons. In Ingersoll's hand power Rock Drills the blow is obtained from the compression of a 200 lb. volute spring.

Borer Bits.—The Bits or Drills which are used with rock cutting machines are in many respects similar to those which are used for hand boring. To keep the bore hole perfectly straight it is necessary that the borer should be well centered, that is a line along the center of the piston rod, if continued, ought to pass through the central edges of the bit. The edge of the bit may be straight or curved. ×, T, and Z shaped bits, give a greater cutting surface, and tend to keep the hole straight. They also clear the hole of accumulating debris better than a straight edged bit. The sharpening of them is, however, more difficult. The shape at the end of the shank will depend upon the method in which it is attached to the end of the holder on the piston rod. That this attachment should be simple and secure is a point worthy of careful attention.

Fixing Rock Drills.—When sinking a shaft the drill is generally supported upon a horizontal bar, along which it can slide, or if desired, be clamped at any point. The bar which ought to be both light and strong, carries at either end a lengthening screw. By turning these two screws outwards the bar can be fixed against the walls of the shaft. Sometimes by means of four radial bars a leg is firmly fixed in the center of the shaft. The top of this leg then acts as a pivot for one end of the stretcher bar carrying the drill, the other end of this bar being fixed by a lengthening screw against the side of the shaft. By this arrangement a large number of holes may be rapidly bored, very little time being lost in shifting the machine from one point to another. Before blasting takes place, the whole of these arms and stretcher bar are by means of a chain raised a sufficient distance up the shaft to escape damage.

When driving a level, a stretcher bar may be used, fixed either horizontally or vertically. In the latter case, to facilitate removal before blasting, it is sometimes carried at the end of a small car running on rails. The drill itself is supported on a small arm branching at right angles from the main bar. This arm can be raised or lowered along the main bar, and it can be fixed to the side of the level by a lengthening screw. If the tunnel which is being driven is of large dimensions, a number of rock drills may be arranged together on a suitable frame-work, the whole being carried on a tram. In open works where it is inconvenient to use a stretcher, the drill may be supported on a tripod.

The speed attained with drills depends a good deal on the speed with which the machines are got to work after blasting. To attain this the chief points to be observed are :—

1. Quick clearing away of smoke by mechanically ventilating the end.
2. Quick removal of stuff by having track laid up to the end, and an organized system of removing.
3. Quick and firm securing of machine in a position for farther work. To do or this, judgment in the men and a good supply of carefully cut wedges are needed.
4. A fair proportion of skilled men looking after the machines.
5. A spare machine or two always on hand in working order.

Economy of Rock Drills.—Rock drills may often be used with advantage in places where the difficulties of transmitting compressed air are not very great, and where the drill being once in position may be employed in boring a number of holes.

The greatest advantage will be obtained when working upon hard rocks like granite, in which holes may be bored at the rate of 5 in. per minute. In these rocks by using a machine, there may be a gain both in time and money. In mild rocks, however, the gain will be only one of time, machine work being perhaps twice as rapid as hand work.

In India (Mysore) the speed compared with coolie hand labour is from three to as much as seven times as great.

In soft rocks hand labour will be the most economical.

When holes are bored by hand as much as 5 or 6 times the quantity of steel may be used, as compared with the quantity which would have been used in replacing old borers had the work been done by machine.

By hand work a man can bore per day in granite 50–100 inches and in limestone 300–400 inches.

Rotary Rock Drills.—In Belgium and France, for boring holes in soft rocks like gypsum. salt or coal, a drill, in principle very much like the tool used by mechanical engineers when boring a hole through an iron plate, is often employed. Lisbet's drill consists of a steel pointed auger turned by a lever and a ratchet, the arrangement being such that it can be either turned round whilst remaining in one position or it can be advanced into the rock.

Macdermott and Williams rock perforator used in making holes for the reception of a hydraulic wedge when falling coal is of this order.

Another form of rotary rock drill is in its action very similar to the large Diamond borer employed in sinking deep bore holes.

In Leschot's drill, a tube is furnished at one end with a crown of diamonds. By suitable machinery this is rotated against the rock which is to be bored. The advance of the borer is effected by means of gearing. This gearing connects the revolving borer with a screw which carries it. As the borer revolves to cut the rock, the screw

slowly turns and keeps it to its work. The rock is cut away in the form of a core which passes into the hollow borer. Water is forced down the borer to clear away the debris.

These diamond borers have not yet been very largely employed for holes for blasting, but the larger drills have been considerably used for prospecting.

Use of Water.—Whilst the boring operation is going on it is customary to keep the hole wet by means of a jet of water. This jet may be supplied from a water tank, the water in which is kept under pressure by means of a pipe in connection with the air conduits of the drill.

By keeping a hole wet, the debris is cleared out, the wear of the drill is lessened and the rate of advance is usually quickened by one half more than what the rate would be if the hole were dry.

ROCK BORING MACHINES.

COMPARATIVE TABLE OF DIMENSIONS (André).

	1 Darlington.	2 Ingersoll.	3 Kainotomon.	4 Warsop.	5 McKean.	6 Sachs.	7 Burleigh.	8 Dubois Francois.
Total length of the machine borer bit not included	36″	30″	36″	24″	38″	23″	51″	86″
Weight of the machine complete without the borer bit	100 lb.	155 lb.	205 lb.	80 lb.	150 lb.	100 lb.	166 lb.	480 lb.
Internal length of the cylinder	11″	11″	15″	8″	7″	8″	6½″	12½″
Diameter of the cylinder	3½	2¾	3½	4	3	2½	2¾	2½
Length of the piston	—	6	8	2½	2	2¾	1¼	3¼
Effective posterior area of the piston	9.62	5.94	9.62	12.56	5 85	4 13	3.54	4.91
Effective anterior area of the piston	4.71	2 80	6.48	5.49	3.93	3.15	1.97	3 14
Maximum stroke of the piston	4	4	4¼	4	4	4	4½	7¾
Weight of the striking mass, piston, piston rod and 8 lb. bit	35 lb.	28 lb.	53 lb.	40 lb.	32 lb.	24 lb,	24 lb.	68 lb.
Volume of air expended in the forward stroke (cubic in.)	38.48	23.76	45.69	50.24	23.40	16 52	15.93	48.05
Volume of air expended in the back stroke (cubic in.)	—	11.20	30.77	21.96	15.72	12.60	8.86	24.33
Volume of air lost on account of clearance space (cubic in.)	14.43	2.80	20.86	15.30	4.99	5.04	1.86	5.16
Volume of air expended for each blow (cubic in.)	52.91	27.76	97.32	87.50	44.11	34.16	26.65	77.55
Force of propulsion with pressure of 30 lb.	182.3 lb.	206.2 lb.	341.6 lb.	416.2 lb.	207.5 lb.	171.3 lb.	140.2 lb.	215 3 lb.
Force of repulsion with pressure of 30 lb.	106 3	56.0	141.4	124.7	85.9	70.5	25.1	26.2
Nature of the feed within	Hand	Autom.	Hand	Autom.	Autom.	Autom.	Autom.	Hand.
Length of feed	24″	19″	18″	—	12″	18″	22″	32″

AIR COMPRESSORS.

An ordinary air compressor may be described as a large air pump. It consists of a cylinder with two sets of valves, one set opening inwards and the other set outwards. By the reciprocating action of a piston, air is drawn in at the former and driven out at the latter.

Blanzy Collieries.—At these collieries the cylinders of the air compressors are placed horizontally. At each end of them there are two valves one of which opens inwards and the other outwards. Through the former, air is taken in from the atmosphere and through the latter it is compressed into the receiver. Round the cylinder there is a jacket in which cold water circulates, and at one end of it there is a water pipe which injects a jet of spray into the cylinder itself.

Colladon's Air Compressor.—In this compressor, which was used during the making of the St. Gothard tunnel, water circulates round the cylinder and at each end two small jets which strike each other and break up into a fine spray are injected. The piston and the piston rod are hollow. The latter, which is continuous, slides upon a small pipe passing into its hollow interior. Through this a jet of water is forced and by means of a series of suitably placed diaphragms a continuous circulation of water is kept up through the piston and its rod.

Sturgeon's Air Compressor.—The action of this compressor depends upon the peculiarity of the inlet valves which allow the machine to be run at high speeds. These valves form a portion of the stuffing boxes through which the piston slides. When the piston moves from one end of the cylinder, the stuffing box by its own frictional grip is carried with the piston until it meets with a stop. This motion of the stuffing box opens the valve, the action being independent of any vacuum being formed inside the cylinder. When the piston returns this frictional grip of the stuffiing box carries it with the piston, and the valve is closed, independently of any compression on the inside of the cylinder. The delivery valves are closed by springs.

The crank of the engine and compressor are placed at right angles so that when the piston of the compressor is near the end of its stroke and the greatest power is required, the steam crank pin is in its best position for supplying the required force.

The receiver forms the bed on which the compressor and its engine rest. By means of a plunger working in this receiver, as the pressure increases the speed of the compressor is decreased.

Ronchamp Collieries.—The compressors at these collieries consist of two vertical iron cylinders in connection with the two open ends of a horizontal cylinder in which there is a reciprocating piston. The vertical cylinders which are partially filled with water, have on their tops a delivery valve working upwards and two clack valves working inwards to admit the air.

By the use of the column of water no clearance space is lost at the end of a stroke, at the same time the air is kept cool even when compressing to a pressure of four atmospheres.

Sarrebruck collieries.—The compressors at these collieries work on a similar principle to those last described, namely by the intervention of a column of water.

Loss of Work.—For the sake of simplicity we will imagine that the compressor is a cylinder C in which there is a piston P without weight, and working without friction. The work which is being done is to compress air through a valve V into a receiver R where there is a constant pressures of two atmospheres.

As the piston P moves down the cylinder, the air, which at the commencement was at the ordinary atmospheric pressure, is compressed, and heat is developed. If this heat escapes through the cylinder, that is to say, if the temperature of the air which is being compressed remains constant, then, if we double the atmospheric pressure, we shall reduce the volume of air in the cylinder C to half its original bulk and the piston will descend half way down the cylinder to M N.

The pressure in the recevier R being equal to two atmospheres, the valve V will now open and the compressed air from the cylinder will enter the receiver. To complete the work which is being done in driving the air which originally occupied the whole of the cylinder C into the receiver R, the piston will work against a pressure of two atmospheres through the space M Q.

If, however, the heat which is developed by compression does not escape, but raises the temperature of the compressed air, and thus causes it to expand, the pressure of two atmospheres will be reached

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when the piston reaches some line like O A—and to force this air into the receiver, the piston, in order to complete its work, will have to work against a pressure of two atmospheres through the distance O Q.

If the volume of air O A Q S forced into R retained the volume with which it entered, there would be no loss of work. But because in practice it is imposible to confine the heat which gives this volume, the air contracts and the actual air in the receiver is not more than if the volumes M Q S N at a lower temperatnre had been forced in. The result of this is that the piston P has worked against a force through the distance O M, which work has been converted into heat and the heat subsequently lost.

To remedy this evil, as it is impossible to prevent the generation of heat, instead of allowing it to raise the temperature of the air it is carried off in the water jacket on the cylinder.

Loss due to Slip and Clearance.—In an air compressor in which a column of water is employed, the water drives all the air through the discharging valves on towards the receiver. Where a piston acts on the air, as it is not possible to allow the piston to come close up to the end of the cylinder in which it works, a certain portion of compressed air always remains in the clearance space at the end of each stroke. This quantity may be increased by the *slip* at the discharging valves, in consequence of which a certain quantity of air which had been once forced out of the cylinder into the receiver, slips back into the cylinder during the discharge. If valves are very large and open wide, this quantity which slips back may be very great.

At the end of each stroke of the piston there will therefore always be a certain amount of compressed air left in the cylinder, and before the inlet valves of the cylinder can open to admit more air, the entrapped air will have to expand until it is below atmospheric pressure. The distance that a piston will have to travel back to allow this expansion to take place, will grow greater as thc compression increases.

Loss of Work from other Sources.—There is a loss of work due to the friction of the various parts of the machine, and also by the friction of the air in the various passages through which it moves. If valves are contracted and have a small lift, there will be

much friction. In the conduits, so long as the velocity of the air is low, say not more than 4ft. per second, the loss by friction is very small.

The friction varies with the area of the surface over which it rubs. It therefore varies directly as the length of the pipes through which it passes, and inversely as their diameters. If the rates of flow rae small, it will vary with the square of such velocity.—(*See Notes on Ventilation.*)

Air Conduits.—These are usually pipes made out of cast or wrought iron, flanged at their ends and fitted together with India rubber washers to make them air tight. If such pipes are long it may be necessary, every 100 yards or so, to insert a bent copper pipe or to bend the pipes to form a loop, so that expansion may take place.

The branch pipes leading from the main conduits up to the machines are usually flexible India rubber tubes, about 2in. in diameter covered with canvas or served with marlin.

Air Receivers.—As it is necessary that a machine should be supplied with a steady flow of air, also as the air is not used so regularly as it is supplied from the compressor, air receivers are employed.

For this purpose old boilers are very often used. These should have a capacity about 20 times as great as the average consumption per minute. As a large amount of water is carried with the compressed air when it leaves the compressor into the conduits leading to the receiver and into the receiver itself, means must be provided for its discharge. In some cases this consists of a simple cock or valve worked by hand. In other cases by taking advantage of the rising of a float in the water, this is made automatic. The exit from receivers should not be opposite to the entrance but by preference on the same side.

The Principal Dimensions of Air Compressors at the Blanzy Collieries (*André.*)

Diameter of steam cylinder	25.6	inches
Diameter of compressor cylinder	21.7	,,
Stroke of pistons	63.0	,,
Area of suction valves	59.0	sq. in.
Area of discharge valves	47.74	,, ,,
Number of revolutions per minute	25	
Velocity of the piston per second	4.37	ft.
Theoretical volume generated by the compression piston at each stroke	13.5	cub. ft.
Which gives for each revolution of the two compressors	54.0	,, ,,
And for the two compressors, running at a velocity of 25 revolutions per minute	1350	,, ,,
This reduced to a pressure of three atmospheres will equal a theoretical volume of	450	,, ,,
In practice the actual volume will not amount to more than 90 per cent. of the theoretical	405	,, ,,

SYSTEMS OF WORKING.

Under this heading, reference is made to the working of openworks, lodes, stratified deposits like coal, and irregular deposits.

OPENWORKS.

Openworks include ordinary quarries for building-stone, slate quarries, the working of alluvial deposits for tin and gold, stockworks, alluvial deposits worked by hydraulic mining, &c.

Among the preliminary questions to be entered into before opening a quarry are,—1 the nature of the material, 2 the quantity of material, 3 the nature of the concession, 4 the cost of raising, 5 the cost of transport.

Amongst other matters determine the existence or non-existence of faults. Is there room for rubbish? If a slate quarry, does the slate decay? Is there water power? Observe the natural *joints*, *posts*, &c. Does the slate split well? Are the slates thick or thin, smooth or rough? Are there *bends*, *cramps* &c.? What is the colour of the slate? Is there much pyrites?

The first work usually consists in the removal of *overburden*, *tirr*, *callow* or *cover*. When removing waste it is contracted for by quantity. Vegetable soil may be removed carefully so that when the operations of quarrying are completed it may be replaced. A clay may be used for brickmaking. Sometimes, as in slate quarries, a large quantity of decomposed slate has to be moved.

All this material should be removed so that it will not interfere with the development of the works.

In attacking a deposit consider,—1. The direction of natural cleavages (*backs* and *cutters*.) It is usual to advance at right angles to the *backs*. 2. The direction of strike. It is usual to advance at right angles to the direction of strike. 3. The direction of dip. If rocks dip steeply towards you, the face of a quarry may be in danger of sliding in upon the works. In such a case a low face is safer than a high one. If rocks dip away from you, water may accumulate in the working

face and further, it is more difficult for workmen to extract a block of stone as compared with the difficulty they experience when the rocks dip towards them. 4. The general contour of the ground. By working up a valley or a river bed, the working face keeps dry by the water draining away from the same.

Lay out a quarry or openwork in *steps*, *stopes* or *galleries*. In some quarries the face of each stope may be a particular bed, so that materials of different qualities may be worked separately.

Slate Quarries.—In opening a slate quarry on a hill side in *galleries*, let each bargain be 10 yards long. Galleries 40 yards long will therefore give 4 bargains. The perpendicular face of a gallery may be 15 yards and its length not less than 20 yards.

Small veins say 25–40 yards thick, worked as underground quarries in Wales, yield the greatest proportion of good slate.

Underground quarries are however more expensive to work. Lights are needed, men cannot distinguish good slate from bad, they are delayed by smoke from blasts, there is danger from falls &c. Beneath a level country engine power is required.

The most favourable position to work is where the cleavage is parallel to the strike, and at right angles to the beds, and the beds dip gently towards you.

To work a bed of slate dipping under a hill from its side, drive an adit from the face of the hill to intersect the slate bed on its under side, and pass through it to its upper side. After this procede to *stope* in horizontal slices across the bed.

If a bed of slate lies beneath level country, sink a shaft on the dip side, drive an air adit along the roof of the bed to meet an air shaft sunk on the rise side, and then work from the bottom of the first shaft with underhand stopes towards the air shaft. A deposit of this description might also be worked as an open quarry. (*See Davies'* "*Slate Quarrying*").

At the Penrhyn slate quarries there are 16 *stopes or stages*, each about 40 feet in height. They are connected by inclined planes; 3,500 men are employed, and in 1872, 250,000 tons of slate were raised. Stopes may be 20 yards broad and from 40 to 60 feet in height. When working horizontally stratified rocks beneath a level country, sink a shaft and as it increases in depth lay off *stopes*. Each of these *stopes*

will advance in a horizontal direction. An inclined wire tramroad will pick up materials from each stage.

The chalk near Paris is worked with chambers and pillars; the lower chambers being smaller than those above.

Slates of Cambrian and Silurian age are found in Wales, the Isle of Man, Wicklow, France, Canada, the United States, Moravia, Silesia. Devonian slates are found in Cornwall, France, Luxemburg. In Devonshire we have Carboniferous slates and in Italy slates of Liassic age.

Irregularities met with in slate quarries are *posts* or *hard bands*, *bends*, *curls*, *dykes*, veins of quartz, hardened ash beds, *joints*.

Usually there is from 10 to 20 tons of rubbish to one ton of slate. In addition to this there may be a large quantity of top rock to be removed at the commencement.

Examples of Openworks.—Most stone quarries, slate quarries like those in Wales, stockworks (massive rock traversed by a network of small veins) like those of Altenberg, Geyer, Zinnwald, and Carclaze in Cornwall. Magnetite deposits of Niji Tagil in Russia. The diamond mines in S. Africa. The Kimberly mine has a face of 500 feet, but there is also much *drifting*. At the Bischoff tin mine in Tasmania, there is a face about 100 feet in height. Alluvial deposits of gold in Siberia, Australia and California.

When a deposit is worked as an openwork there is a saving in the price of getting, timbering, stowing and lighting. Further, all the material is extracted. Callon reckons the saving in excavating coal at 27d. per ton or $27 \times \frac{3}{4} = 20\frac{1}{4}$d. per cubic yard.

This sum must be calculated for any given seam, and before deciding to work by removing overburden, the result must be compared with the cost of excavating from the surface.

The disadvantages of openworks are the destruction of the surface and the expense of removing the overburden.

Monoliths.—One of the largest blocks of stone excavated by the Egyptians is 99 feet long.

Cleopatra's needle, now in London, is a piece of syenite 64 feet long and 8 feet square at the base. The Obelisk of Luxor, now in Paris, is 79 feet high and 8 feet square at the base.

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Alexander's column in St. Petersberg is 84 feet long. When excavated it was 102 feet long and 14 feet square at the base.

Monoliths like those met with in Egypt may have been obtained by first uncovering two faces of the rock and then adopting one of the three following processes.

1. By simultaneously striking a number of wedges inserted along a line parallel to a vertical free face.

2. By cutting a groove on the horizontal free face parallel to its length. Covering this with fire to heat and expand the rock, and then causing sudden contraction to take place by the application of water.

3. By driving plugs of wood into holes along a line parallel to the length of the block, and then causing the same to expand by the application of water.

In Siberia when simply breaking down rock, advantage is taken of the intense cold during winter, when fires are lighted against the working faces which by subsequent contraction during cooling are split and fractured.

HYDRAULIC OR PLACER MINING.

In Nevada County, California, the writer saw deposits which were upon the face from 100 to 300 feet in height. They consisted of gravels, sometimes cemented to form a hard mass, tuffs and clays. When deposits are strongly cemented a tunnel is driven a distance two-thirds of the height of the bank to be blasted and terminated by cross drifts. At the end of the cross drifts from 100 to 1,000 kegs of powder may be placed (1 keg=25 lbs.) and then exploded. From 10 to 20 lbs. of powder are required per 1,000 cubic feet ground. The talus thus formed is washed away by jets of water.

Dams, Flumes, Pipes.—The water supply is contained in dams. Earthen dams ought not to be less than 10 feet thick at the top and if over 60 feet in height they are dangerous. Wooden dams may be built of a series of cribs each about 10 feet square the face of the dam being made tight with a skin of planks. Dry rubble stone work faced

with planks is also used for dams. From the dams the water is conveyed in open ditches and wooden flumes. Because the latter are smaller than the former, they have a steeper grade.

Ordinary flumes made of 2½ inch planks, may be 3 feet 10 inches broad and 1 foot 10 inches deep. Some flumes have dimensions 1 foot by 4 feet. The grade is usually about 30 feet per mile.

In a commanding position above the claim a pressure box built of 1½ inch planks is placed, from which the water is conveyed in iron pipes down to the claim. These pipes may be from 8 to 40 inches in diameter and may have to stand a pressure of 400 feet.

Branch pipes may be made of crinoline hose which will bear 180 feet of water, From the end of these the water is allowed to escape through a nozzle from 3 to 11 inches in diameter. The pressure is from 100 to 200 lbs. on the square inch. The material washed away passes along trenches lined with boulders and blocks of wood about 1 foot 6 inches square, with a space between each of 2 inches. It is here that the gold is collected.

The water, is supplied by companies (Ditch companies) at a certain rate per miner's inch. A miner's inch is the quantity of water which will escape though a hole 1 inch square in a 2 inch plank under a head of 6 inches over the opening, in 10 hours. The miner's inch however, varies in different localities. The definition here given is however the most usual. The discharge of the miner's inch per hour is 94.404 cubic feet. The cost of working is 2 to 4 cents per ton. Under favourable circumstances gravels containing 3 cents worth of gold per cubic yard will yield a profit.

When working in an excavation on a flat country it is found that the debris and water from the face can be raised to the ditches on the general level by a heavy jet of water shooting up an inclined pipe.

The method of working destroys the surface of a country, silts up rivers resulting in floods, interferes with harbours, and creates other damage. In 1871 in California there were 516 ditches bringing water, having a length of 4,800 miles.

One company with 10 men at 16/– per day and 4,000 gals: of water worked 224,000 cubic feet of gravel in 6 days. The gold obtained was valued at £600. The profits at £470. Another company with 40,000 gals: of water in 100 days washed 1,000,000 cubic yards and obtained £6,400 worth of gold. Profit £2,400.

For more complete details see Bowie's "*Treatise on Hydraulic Mining.*"

WORKING LODES.

Before opening a metal mine it is well to consider the following points.

The nature and quantity of the material to be extracted; the general form of the deposit; the depth at which you may have to work; the nature of the boundaries (*roof*, *floor*, *walls*) the thickness of the deposit; its hardness and physical structure; the existence or non-existence of faults; the quantity of water which may be met with &c.

To work economically, if possible arrange the works so that the underground transport shall be short, the *stopes* clear of water, and so that there shall be good ventilation, and the men not crowded.

In ordinary lodes, half or three quarters of the men employed underground may be engaged in exploratory work (*dead* or *tut* work.)

Working Lodes of Ordinary Width.—Ordinary lodes which are from 2 feet to 6 feet wide, are worked either by overhand or underhand stoping.

In Cornwall the price of *stoping* varies from 2/6 to 6/– per ton.

1. Overhand Stoping.—It is better to commence from a winze than by breaking in. Tools and materials can be easily lowered down a winze, and the winze also acts as a passage for air. A fair height for a stope is 6 or 7 feet. A high face allows the material to come down in blocks, but the stage necessary for the workmen in such a case has to be removed before firing a shot. The length of a face may be from 20 to 40 feet. When the stopes are small the men may be inconveniently near to each other when a shot is fired. The workmen advance quickly if the joints in the rock dip towards them.

A *pass* is formed by the walls, stemples and attle, or it may be timbered like a winze. The bottom of the pass ought to be larger than the top to prevent jamming. In a 90 feet pass the top may be 3 feet square and the bottom 4 feet 6 inches square. The limiting inclination is about 50°.

The advantages of overhand stoping are,—1. The ground breaks readily. 2. The ore is thrown down a pass. 3. The water drains

away. 4. The ventilation is good. 5. There is less timber employed as compared with underhand stoping, unless there are not enough *deads* to fill in with.

The disadvantages are.—1. The dead work of driving a level beneath the deposit. 2. The danger to the workmen. 3. The loss of ore amongst the attle. 4. The inconvenient position in which the men work.

2. Underhand Stoping.—The floor and face of a stope ought to dip downwards. The height of the face may be 6 or 10 feet and the floor 25 to 40 feet. The advantages of this method are,—1. The work may be commenced without previous dead work. 2. There is little risk of accident. 3. The workmen may use heavy double handed tools. 4. There is little loss of ore.

The disadvantages are,—1. The passing up of the ore. 2. The risk of water accumulating. 3. The quantity of timber required. 4. The difficulty of ventilation.

These latter objections hardly apply if the usual system is followed of putting down a winze and bottom-stopping into it.

Narrow Lodes.—In working narrow veins a strip of the wall may be first removed and then the vein broken down unmixed with wall rock (*stripping*), otherwise they are worked by the ordinary methods of stoping.

Working Wide Lodes.—Wide lodes are lodes where the material of the lode is too weak to stand as a roof without support. Generally any lode over 8 or 10 feet will fall within this category.

1. By Horizontal Slices.—A level is driven along the footwall and at intervals of say 18 feet, levels or cross cuts are driven across the lode to the hanging wall. By driving these levels, a certain quantity of material is obtained from the lode. The cross cuts are then packed with attle, and a second set of new cross cuts are started inside the pillars: These being packed, a third set of cross cuts (each cross cut being 6 feet in width) will complete the extraction of a horizontal slice taken out transversely or across the lode from the footwall to the hanging wall. A second slice is then taken out at a higher level and so the work proceeds slice by slice working upwards.

If it is not desirable to come in contact with the stowing, a thin slice of the vein may be left as a floor between two successive slices.

When taking out slices in a descending order, the works in the upper slice should be in advance of those below.

2. By Vertical Slices.—At Almaden in Spain, slices are taken vertically from level to level, the walls being supported by masonry.

3. Working by Pillars.—This method is adopted where stowage is difficult to obtain. The slices, which are much thicker than in the previous method, are removed in descending order. Cross-cuts are driven from a main level in the hanging wall side, and possibly also another set of levels at right angles, so that a slice is divided up into a number of pillars. The pillars and portion of the roof are then cut away until the roof commences to fall, these operations commencing in the part of the mine furthest removed from the shaft. After this a second slice is commenced at a lower level, leaving a sufficiently strong roof between it and the crushed-in workings above. When working several slices simultaneously, the stripping of pillars, in a lower level must not be commenced until the upper level has been completely worked.

Examples of the cross-cut method with pillars are found at the Stahlberg Iron Mines, the calamine workings in Silesia, the alum shales at Liége &c.

WORKING ROUND HORSES OR IRREGULAR DEPOSITS.

If the deposit is large, drive round it a horizontal contour level. This may be used as a main level from which cross cuts may be driven and the deposit worked out by the method of horizontal slices. The pillar method may also be employed. When material is extracted by a series of chambers the pillars of a lower tier are made larger than those of a tier at a higher level. At Musen near Siegen the vaults are 18ft. high and the pillars between are 12ft. thick. Only one-third of the material (iron carbonate) is obtained from each level.

At the salt mines of Salzburg the salt is obtained by dissolving it out of chambers.

WORKING STRATIFIED DEPOSITS.

As a typical example of a stratified deposit we will take coal. The more ordinary conditions which may affect the method chosen to work a given seam of coal are the following.

1. The thickness af the seam.
2. The dip of the seam.
3. The nature of the floor and roof.
4. The depth of the seam from the surface.
5. The chemical and physical nature of the coal.
6. The natural cleavage or cleat of the coal (*backs* and *cutters*.)
7. The presence or absence of water.
8. The vicinity of other seams or workings which should not be interfered with.
9. The position of the seam with regard to rivers, roads, canals, towns &c.

Before commencing work, many commercial questions as to the price of labour, the means of transport, facilities for sale &c. must be answered.

The following classification of systems of working is modified from a table given by O. J. Heinrich.—(Trans. A. I. M. E. Vol. II. p. 107).

I. For Seams of 10 Feet or less.

1. Dip from 45° to horizontal.
   - *a.* Pillar and stall, post and stall, stoop and room, pillar and chamber (*Exploitation pas serres hautes et courtes. Kurzer and langer Pfeilerbau; Felderbau*).
   - *b.* Long wall system (*Exploitation par tailles grandes, droites et couchantes; Strebbau mit Breitem Blick; Stossbau; Diagonaler Pfeilerbau*).
2. Dip from 45° to vertical.
   - *a.* Overhand stoping (*Exploitation par tailles à gradins renversés; Firstenbau*).

b. Working by longitudinal pillars (*Exploitation par serres longitudinales; Streichender Pfeilerbau*).

II. **For Seams over 10 Feet Thick.**

1. Dip from 45° to horizontal.

   Post and stalls with or without stowages (gobbing) (*Exploitation par serres à methode par remblais; Pfeiler und Felderbau mit oder ohne Bergversatz*).

2. Dip from 45° to vertical.

   By crosscuts or benches and stowage (*Ouvrage à travers et par remblais; Querbau mit Bergversatz*).

All these methods are variations of the post and stall method, the long wall method, or of the methods described as applicable to the working of lodes which may be regarded as inclined seams. In the following description of methods, the above classification is not strictly followed.

a. Post and Stall Method.—Usually a shaft is sunk on the deepest side (dip side) of the area to be worked, in a central position on that side.

Two or more levels (say 10 feet wide) are driven parallel to the strike (*dip head level*, *lodgment level* or *water way*) slightly inclined towards the shaft. The lower of these is used to convey water to the sump, while the others are employed for haulage or as a level from which the stalls or bords are laid out when working (winning) the coal. These are usually laid out, so that they advance in a direction at right angles to the cleat or cleavage, such a direction being known as "*on the face or bordways.*" The direction at right angles to this is known as "*on the ends.*"

Occasionally circumstances (as for example a steep dip) demand that the boards shall advance obliquely to the cleat or even parallel to it, that is on the ends.

When there is only one direction of cleat, the coal has a tendency to break into prismatic forms. If there are two sets of cleavage at right angles (backs and cutters) the coal will be cuboidal—if oblique, rhomboidal &c If advancing at right angles to well defined backs, the coal will come away with ease, but in consequence of the under cutting and falling, so many additional fissures may be produced, that

two much *smalls* or dust coal may be produced. In such a case, advance the bords in some direction oblique to *the backs* or even *on end*.

The *winning headway* from which bords are driven may therefore be parallel to the strike, the dip, or in some intermediate direction. At least two bordways which are as wide as the roof will stand without support (10 to 15 feet) are driven together. After advancing a certain distance the working faces are joined by a passage or *thirling*, parallel to the *winning headway*. The bords are then advanced, a second *thirling* driven and the first *thirling* may be temporarilly stopped. The ventilating current now passes round the working faces through the second *thirling*.

By a system of bords and thirlings at right angles to each other the area to be worked, is cut up into a series of pillars.

If these pillars are made too small they become crushed and cannot be subsequently removed, and more than half of the whole seam may be destroyed.

In getting iron ore in headings, the cost is 1/– to 2/6 per ton while when robbing the pillars it is 9 d. to 1/6 per ton.

Dimensions of Pillars.—The dimensions of pillars depend upon the strength of the roof, the floor, and the coal, the depth below the surface, and the length of time they have to stand. Thus with a soft *floor* or *sole* or with soft coal, large pillars are required. Hyslop gives the following table :—

Depth in fathoms.	Size of pillars in yards.	Proportion of pillars to bords.
20	20 × 5	.41
40	20 × 6	.50
60	22 × 7	.52
80	22 × 8	.57
100	22 × 9	.59
120	22 × 12	.61
140	26 × 13	.63
160	26 × 16	.66
200	28 × 16	.71
260	30 × 21	.77
300	30 × 24	.79

An average size is 12×40 yards. With long pillars the proportion of length to width is 4 : 1 to 3 : 2. If pillars are too long there is a difficulty in ventilation. When a pillar is too small or too weak to support the superincumbent strata, it gives way by cracking and crumbling. This is called *thrust*. "*Sits*" are falls of the roof. When the pillar is small and the floor is soft, the downward pressure causes the floor to rise. This is called *creep*. When *creep* once sets in, it has a tendency to spread and the expense of repairing roadways and keeping airways open, may be very great.

Near the shaft, large pillars at least 40 yards square are left to support the shaft. This may be sufficient for a depth of 100 yards. For each increase of 20 yards in depth the sectional area of the *shaft pillars* may be increased 5 square yards.

Between main roadways and areas which are being worked, a strong barrier of coal is left to protect such roadways.

An area may be worked as follows:

1. As a whole, commencing in the vicinity of the shaft and working *outwards* or *away from home* until the limits of the area are reached. After this the pillars are removed (*robbed*) and the roof collapses. The pillars first formed are therefore worked the last.

In this system many air ways and roadways have to be kept up for many years and the risk of thrust and creep and other accidents is increased. The pillars first formed have deteriorated by long exposure to pressure and to the atmosphere.

2. By driving roadways to the limit of the area and working towards the shaft ("*towards home*"). In this way pillars may be removed shortly after they have been formed.

3. By Mr. Buddle's system of dividing a district into *pannels* or small areas separated from each other by strong barriers of coal, each 40 to 60 yards wide. Each of these pannels is worked by its own system of bords and pillars, has its own ventilation and is independent of neighboring pannels.

4. By long block system. Long pillars may be 120×60 yards. Examples are the "Single Road" and "Double Road Stall" systems of S. Wales.

Getting the Coal.—The first operation is to cut a groove in

the coal at the bottom of the seam parallel to the floor, to a depth of 2′.6″ or 3′. This is called *holing*, *kirving* or *benching*; and is performed by *hewers*. The height of the groove at the entrance will be 9 to 12 inches.

Each hewer works a certain length called a *stint*. Small props called *sprags* are used to prevent an unexpected fall of the undercut coal. The next operation is *shearing* or the making of vertical cuts. The last operation is the removal of the *sprags* and the forcing down of the coal by wedges or blasting. This is *getting* the coal. Lastly the coal is loaded in *sleds* or *trams* and taken to the nearest roadway where *putters* take it to the bottom of the shaft.

b. Long Wall Working.—From a shaft sunk on the dip side of the seam two roadways with a good wall of coal are driven waterways. The face of the upper road is undercut along its length and the coal is got. After a certain number of strips have in this way been removed the roof beind the miners falls to form *goaf*. To prevent the roof falling too near the working face, one, two or three lines of props at from 3 to 9 feet apart are placed parallel to the working face. After a strip of coal has been got the back row of props are removed and are reset in front of what was the front row. On the top of the props a *cap* or *lid* 2 to 3 inches thick and one or two feet square may be placed. Roadways or gob roads are built through the fallen roof or *gob* to connect the working face with the main roadway leading to the shaft. Along the sides of these roadways, pack walls are made of refuse stones obtained from the coal or fallen roof.

Instead of having one long wall (working face) from 100 to 500 yards long, a number of working faces (stalls) arranged in a step like form and each about 50 yards in length may be used. Each of these will have its gob road leading to a main roadway. Such a method would be used in a steeply inclined seam. It is evident that instead of having the working faces parallel to the strike, they may be in some direction depending on the dip of the seam, the direction of *cleat*, &c. In working homewards, gob roads have not to be built and much expense is saved. There is however the preliminary expense of driving roadways through the solid coal to the limit of the area to be worked.

The long wall method gives, according to Greenwell, about 14 % more coal than the post and stall method. Coal cutters may be most advantageously employed, there is little risk of danger, the ventilation

is good, and the working can be easily supervised. It is best applied to seams of moderate dimensions.

For thick seams this method requires long props which are expensive and the gob roads are expensive. In all cases the work must go on continuously and regularly. The system is interfered with by faults.

Long wall has been used at Monkwearmouth at a depth of 1,800 feet on a seam 6 feet thick. It is employed, in Staffordshire, Derbyshire, Leicestershire, Somersetshire, &c. In Lancashire it was employed in 1812. At Zwickaw in Saxony they work homewards. Other examples are in France, Westphalia, Japan, &c. In America hard coal is sometimes worked by long wall, while soft coal is worked by post and stall, one reason being that the *smalls* of soft coal can be better utilized than the *smalls* of hard coal.

Effect of Subsidence.—The removal of a seam 2 or 3 feet thick will usually produce but little effect at 30 feet above it, but the removal of a 20 or 30 feet seam may produce an effect at 100 feet. In the case of 2 seams near to each other, it will usually be best to work the upper one first.

In America two parallel seams have been worked simultaneously by the post and stall method, the pillars in the lower set of workings being exactly below those in the upper set. To carry this out practically is a matter of some difficulty.

The subsidence on the surface follows the workings beneath but at a distance back from them dependent on the depth of the workings, the nature of the strata, the dip &c. R. Hausse gives the following rule to determine the direction of the plane of fracture making an angle θ with the horizontal plane, the dip of the strata being β.

$$\tan \theta = \frac{1 + \cos^2 \beta}{\sin \beta \cos \beta}$$

Callon says subsidence causes fractures along the perimeter of the area at right angles to the plane of stratification.

c. Seams which are steeply inclined.—If a seam is steeply inclined and the advance is towards the rise, then there is danger of coal sliding from the working face down upon the men. This may be partially obviated when working by a long wall method,

by making the working faces (*stalls*) short and arranging them in a ziz-zag line—or by advancing in the direction of the strike by a system of overhand stopes as in the north of France and Belgium.

The steeper the dip, the smaller are the stopes. Stopes are usually about 6 feet high and 12 feet long. Inclined planes or passes through the goaf carry the coal to a lower level. With a dip of 30° to 60° the stopes or step like stalls may be as much as 50 by 70 yards.

d. Thick Seams (*The Staffordshire Method*).—Square work. This method which is applied to the 10 yard seam at Dudley is a form of post and stall. Narrow levels are driven at right angles, dividing the coal up into blocks 50 to 70 yards square. Narrow holes (*bolt holes*) 8 or 10 yards long are driven into these blocks, after which the work goes on in stalls to the right and left until the block has been hollowed out to form a chamber, the roof of which is supported by four or more pillars. These which are 3 or 4 yards square and 12 yards apart are called *men of war*. The men work upwards toward the roof in overhand stopes, standing on the broken coal or scaffoldings. The final operation is to thin the pillars until the roof falls in, when the chamber is abandoned and the *bolt holes* are closed. The method is dangerous and the loss of coal is great say 50 per cent.

Some of the Pensylvanian anthracites are worked in chambers, but by commencing at the top and using underhand stopes.

e. Working in Slices (*Blanzy*).—Work off an upper slice say 6 feet thick of a thick seam by post and stall or by long wall and allow the roof to settle. Next a second slice may be taken beneath the broken roof, and so on until the seam has been completely won.

At Montceau les Mines, the seam was divided into 3 parts by two partings each 3 m. thick the coal between being 1.5 m. The upper layer was worked with pillars (12 m. in direction of strike and 25 m. in direction of dip). These were cut away until they were 4 m. × 5 m. when the roof crushed. Two years later the lower layer was worked.

In some mines it has been found that it does not require a great interval of time for the broken roof to consolidate, so that several slices may be practically worked together, providing that the upper workings are kept in advance of those below.

If packing is employed, a seam may be worked in slices taken in ascending order. In this way gob roads are under foot instead of over-

head. In this case the coal settling on the packing may be much fissured, and the risks of spontaneous combustion are increased.

If a seam does not exceed 15 or 16 feet, it may be worked by overhand stopes, props being used to support the roof and the faces of the stopes.

If a seam is thick and has a steep inclination, it can be worked by a system of cross cuts with stowage, taking transverse slices in an ascending order. The galleries may be driven in the wall. (See method of working wide lodes.)

As Dombrowa Poland, seams 52 to 59 ft. thick, and inclined at 15° to 40° are worked by cross-cuts in descending order, a layer of coal being left between each slice. In Upper Silesia seams 33 ft. thick are worked in long blocks paralled to the strike. At Kladno in Bohemia the workings are somewhat similar to those in Silesia, but here they bring down 20 or 30 ft. of coal at one fall.

Cost of Breaking Ground (*Agendas Dunod*).—For driving a level 3.50 m. square section (2 m. high with a circular crown and 2 m. broad), miner's wages 4 francs per day, powder 2.50 francs per kilogramme, 2 shifts per day and 25 working days per month. See the following table.

Nature of Rocks.	No. of days per running meter.	Powder consumed.	Price.		Monthly advances.
			Per running meter.	Per cubic meter.	
Exceptionally hard rocks.....	50	Kilos. 12	Francs. 236	Francs. 67	Meters. 2
Hard granite and quartzoze rocks.	20 to 30	8 to 10	120 to 145	34 to 41	3.33 to 5
Hard veins and very quartzoze..	24	3.5	140.75	30	4
Hard coal measures, grits &c.	15 to 20	4 to 8	70 to 100	20 to 28.50	5 to 6.87
Moderately hard coal measures ordinary granite	10 to 15	3 to 4	40.50 to 70	13 50 to 20	6.67 to 10
Ordinary coal measures, soft argillaceous or micaceous schists.....	7 to 10	1.5 to 3	31.75 to 47.50	9 to 13.50	10 to 14
Soft coal measures	4 to 6	1 to 1.5	18.50 to 27.75	5.20 to 8	15 to 25
Coal 6m. sq. in section.....	2 to 4	0.0 to .3	8 to 16.75	1.33 to 2.80	25 to50

In Cornwall the cost of sinking per cubic fathom is given by Collins as follows:—

	Near Surface.	Below 20 Frm.
In soft killas or clay	£ 2 to £ 3	£ 3 to £ 4
In compact killas or pick & gad ground	£ 4 to £ 6	£ 5 to £ 8
In fair blasting ground	£ 6 to £ 20	£ 10 to £ 30

For driving levels; ⅓ cheaper for non-blasting, and ½ cheaper for blasting ground. As Bottalack the cost of driving levels 3′×6′ per running fathom is from £ 4 to £ 6.

In the Polberro mine the cost per fathom was for shafts, £4.12. for driving in the lode £6.8.6 and for stoping £2.2.

For a level 7′ high, 3′.6″ in the cap, 4′.6″ at the bottom with half timber sets and 1½ inch laths, the cost of driving per fathom will be about £2.10 to £3 and for timbering £1.5 to £1.10 (*Collins*).

The following numbers give the relative cost of different kinds of work, stoping openworks being taken us the unit.

Driving small levels	4	Stoping underground	2.6–3
Large sections	1⅓	,, moderate vein	2
Vertical rising	6	,, thick vein	1⅓–2
Shaft sinking	6-8		

In England under fair conditions, two men and a boy in a shift of 8 hours, may extract 8 to 10 tons of coal. Hewing and falling may cost 8d. to 10d. per ton, but including proping, cogging and stowing 1/6. per ton for large, and 6d. per ton for smalls.—(See cost of sinking p. 108.)

Waste of Coal by Weathering.—The weathering of coal greatly depends on the absorbtion on oxygen forming CO_2., H_2O. and upon the presence of Iron Pyrites (Marcasite). By weathering, the heating power may be diminished 6 % or more, and the coking power 2 to 5 %. The volatile matter in coal rapidly diminishes by exposure. Pyrites decomposes rapidly in a damp place. Coals without pyrites may decompose more rapidly in a dry place.

Fires in Coal Mines.—Some coal mines have continued burning for 50 and some even more than 100 years.

Fires are occasioned by carelessness. Timber near underground boilers has taken fire. The material in stables has been accidently

fired. Ventilating furnaces have not been carefully protected. Coal dust in suspension in the air may be exploded by a flame and a fire result &c.

A common cause is spontaneous combustion, which may originate in the coal or in a heap of waste material like greasy cotton.

Coals with a large percentage of oxygen are liable to spontaneous combustion. Burat says that poor coals giving a long flame are subject to spontaneous combustion.

In many coals hydrocarbons like CH_4. are formed (see Ventilation). By a fall in atmospheric pressure or by a fall of the roof a large quantity of these gases may be forced into a level and coming in contact with a light, the result is an explosion and fire.

If the pillars of a mine are too small, they are cracked and fissured and the surfaces for oxidation are increased. The decomposition of Iron Pyrites also tends to crack the coal and in this way assists the oxidation. The result of oxidation is that a pillar may become hotter and hotter until fire is produced.

Preventing Fires.—Careful inspection. Walling off places where gases accumulate. Burying places which are burning, with sand. A small quantity of water producing dampness, may increase the chemical changes which are producing heat. Keep the workings clear of small coal. Good ventilation sometimes may cool a place, while at other times it may accelerate oxidation.

Extinguishing Fires.—Certain American mines have a water supply through the main levels under the pressure of a main in the shaft. By a hose which may be screwed on these pipes, a jet of water may be quickly turned upon a fire in any portion of the mine.

Portable *Extincteurs* giving a jet of CO_2., and water, have sometimes been useful.

Fires have often been extinguished by damming so that no fresh air can gain excess.

To approach a fire, run a central brattice along a level, along one side of which the smoke escapes, while a current of air is sent in upon the other side.

The ordinary plan is to drown the mine with water (which may partially destroy the timbering, cause falls &c.,) steam, or carbon dioxide.

Carbon dioxide was used successfully at Ruabon.

265 lbs. of limestone and 580 lbs. of ordinary hydrochloric acid which may be mixed in a box lined with lead give 1,000 cubic feet of gas. The same gas may be produced by burning coke or coal together with limestone in a specially arranged furnace. The resulting gases (CO_2 and N) are forced into the mine by a jet of steam.

Officers and Workmen in Mines.

In Collieries.				In Metal Mines.
Chief Viewer	corresponds to			Inspecting Captain or Manager.
Resident or Underviewer and Overman.	}	,,	,,	{ Chief Resident Captain, Sub-captain, Grass-captain.
Deputy Overman.	nearly	,,	,,	Pitmen, Shaftmen.
Wastemen.	,,	,,	,,	Timbermen.
Firemen.				.....
Hewers, Getters.		,,	,,	Tributers and Tutworkmen.
Putters, Rolley drivers.		,,	,,	Trammers.
Onsetters, Hangers-on.		,,	,,	Fillers.
Doorkeepers, trappers.				.....
Banksmen.		,,	,,	Landers.
Screeners.		,,	,,	Dressers.
Smiths, Carpenters.		,,	,,	Smiths, Carpenters.
Enginemen.		,,	,,	Enginemen.

In *Tutwork*, *todtwork* or *dead work* when driving levels, the price paid is usually so much per fathom.

Tributers work an area called a *pitch* and receive a certain percentage of the value of the ore extracted. If ore is rich say 1/– in the £, but if poor 13/– in the £. These contracts are usually let to the lowest bidder. Each class of workmen has certain charges to pay. Thus a tributer may pay 1/6 per ton for drawing and crushing the ore, 6d. per ton for dressing, 2/– for sampling, 8d. for weighing, mending chisels 1d. each, mending 3 picks 2d. candles 8d. per lb, also he pays for powder, fuse, doctors fund &c.—(*Collins*).

As Himmelsfurst, Saxony, there were employed 1 surface captain, 1 underground captain, 16 under captains, 362 miners and hewers, 54 timbermen and masons, 122 trammers, windlassmen and fillers, 171 dressers, 21 smiths, 3 enginemen, 2 watchmen and 1 clerk. Total 754.

At the Levant Mine, Cornwall. There are about 300 people underground and 100 on the surface.

At the Bottalack Mine, there are about 190 underground, and 210, including boys and girls, on the surface.

At the Whitehaven Colliery about 1,500 men and boys are employed.

At the Ashio Copper Mine in Japan 4,000 workman are employed.

SHAFTS.

Varieties of Shafts.—Engine, ventilating, pumping, coaling, upcast and downcast shafts. Blind shaft, winze, sump shaft, broken shaft.

The Adalbert shaft at Prizbram 572 fms. The Viviers Reunis coal mine at Gilly near Charleroi is 581.5 fms. (the staple at the bottom being 11½ fms.) In England the Ashton Moss Colliery near Manchester is 475 fms. The Dolcoath tin mine is 404 fms. In 1886 the Comstock reached 3,250 feet.

Site of a Shaft.—Place a shaft on level ground or in a valley near the center of the works. Works however are usually located to suit the shaft and not vice versa. A shaft should not be on swampy ground or where it may be flooded. Shafts near the sea have been lost by tidal waves. It ought not to be too near a building or too far distant from a road or canal. It must be remembered that the engineers require a water supply.

The mouth of a shaft should be 20 feet above the surrounding level. This enables a *bank* or *stage* to be constructed from which the material from the mine may be readily discharged.

Shafts are usually sunk from the surface but as in the case of long adits, which to expedite the work may be commenced at several points along their length, a shaft may be sunk from the surface and from several levels, the manner on which the different holings meet being dependent in the accuracy of the mine survey (Ex. Königin Marie Shaft, Hartz Mts).

In coal seams, sink on the dip side of the bed and on the dip side of an ordinary fault. With a reversed fault or where the seam dips against the fault, it may be better to sink on the rise side.

A shaft may be vertical or inclined, in a lode, or outside in the country rock. Its long side is placed parallel to the strike of vein. Vertical shafts are usually sunk on the hanging wall or dip side of a lode, to intersect the lode at about half the maximum depth of the intended workings. This arrangement gives a minimum length for

the *cross cuts*. Such a shaft does not explore the lode or help to pay the expense of sinking. It is good for a quick out-put and saves expense in winding, pumping and (for shafts of equal depth) in timbering.

If a lode is nearly vertical, the shaft may be sunk entirely on the foot wall side. There is less chance of dislocations taking place in the walls of the shaft. A rectangular shaft ought to be sunk with its long sides transverse to the cleavage or bedding of the rocks through which it passes; this may prevent the sides from slabbing off. It is assumed that the plane of bedding is nearly vertical.

An inclined shaft in a lode is usually cheaper to sink, and it may be drier than a shaft in the country rock. While sinking, the lode is explored and ore is won. If it is in the country rock it ought to be on the foot wall side of the deposit. It is longer than a perpendicular shaft, and there may be more movement in its walls. For these reasons more timber and other materials are required. Mechanical arrangements may be complicated and wear and tear is increased. There may be difficulties in opening parallel veins near such a shaft.

Form and Size of Shafts.—Rectangular, square, polygonal, elliptical or circular.

Rectangular shafts are common at metal mines; the two long sides being formed by the side of the lode, the two ends being portions of the lode and forming shaft pillars. On the Comstock lode the *end-pieces* are 5 to 6 feet, while the *wall plates* forming the long sides, are from 20 to 24 feet. More ordinary dimensions are 4 feet to 10 feet by 9 feet to 16 feet. Rectangular shafts are convenient for dividing.

Square shafts are stronger than rectangular shafts and require less material for a given sectional area.

Circular shafts are usual at nearly all coal mines, they are the strongest, most easily made water tight, give the largest area for a given periphery, but they are not so convenient to divide. A common diameter is 12 or 14 feet but for *costening* 4 feet may be sufficient.

Shaft Timbering.—Shafts in lodes sometimes only require timbering on their shorter sides, the longer sides being the walls of the lode. This timbering consisting of *sets* and *laths* is very similar to that used for the roof of levels.

Timbering a shaft 10 ft. by 7 ft. with *sets* and *laths*, the sets of 9 inch timber 4 feet apart, the laths 1½ inch thick, costs about £3.10 per fathom of depth (*Collins*).

Prop-crip Timbering.—On the surface a *day frame* with bearing ends is placed. After excavating 8 or 9 feet, bearing stemples are wedged across the shaft from the hanging to the foot wall. On these a *crib* is laid, and on the crib 4 props which carry a second crib and so on until the last props bear against the under side of the day frame. Wedges are driven in between the cribs and the sides of the shaft. The bearing stemples may bear and rest on blocks. The distance apart of cribs may be 2 feet 6 inches to 4 feet. Lagging may by used behind the cribs.

If the wall plates of cribs are liable to be bent, they may be propped by *cross bars*, *buntons* or *dividings*, and to prevent cribs being forced out of shape, inclined struts or stringing planks are used. Stringing pieces may be notched against the cribs and held in position by cross bars, or spiked,

When one frame or crib is placed upon another the lining forms solid crib timbering.

Frames are kept apart by *corner pieces*, *studdles* or *jogs*.

Plank or Box Timbering.—This consists of planks 6 inches or 8 inches deep and 2 inches to 4 inches thick, notched together to form a box; the ends of the sides of which may be left projecting.

These boxes are supported by their ends or on stemples, and they are packed behind with attle or clay. This timbering is only suitable for small shafts.

Circular shafts may be lined by planks bent to form hoops with lagging behind or with wooden cribs built up in sections and propped apart. These are lined on the inside with planks, on the inside of these again lighter cribs may be placed. Where the pressure is great, the cribs may be 9 inches × 7 inches.

In sinking, a depth may be reached equal to the length of the lagging, say 9 feet. Here a crib is placed and the lagging placed behind it. A second crib is then placed on props (*punch props*) at a height of 3 feet in front of the lagging, and so on up to the surface. The excavation now proceeds for another 9 feet parallel with the

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inside face of the crib first inserted, which may be supported by rods, chains or *stringing deals* from the surface,—by driving in bars beneath it,—by props from below, or by a portion of the earth or rock beneath it, this being removed gradually.

Solid Crib Timbering.—Where pressure due to a head of water has to be resisted, one solid crib is placed upon another. The lowest in the series is usually a *wedging curb* or *crib*. The cross section of the timber out of which it is built is greater than that of the cribs it carries. Between it and the walls of the shaft there is a plank sheathing which is forced back against a bed of moss by a series of wedges—the moss being between the sheathing and the walls.

Sinking in Soft Ground.—In soft ground the method is similar to *spilling*, yet to be described. By driving diverging *spills* the section of the shaft remains constant (*Dutch cribbing* or *framing*). If the *spills* are driven vertically, the diameter of the shaft is decreased at each length of timbering by an amount equal to twice the thickness of the *cribs* and *spills*.

Another method, by which a short length of soft ground may be passed when descending from the surface, is to make a large conical excavation, timber or brick vertically upwards and then fill in.

Soft ground may be penetrated by sinking a lining or drum curb formed by a succession of cribs or tubs provided with a cast iron shoe. As the lining descends,—being forced downwards by screw presses or weights, its length is increased above and the shaft excavated.

A great difficulty is to keep the drum vertical, especially when obstacles like boulders are met with.

For details and drawings relating to the above notes see "Mine Timbering and Shaft Timbering" by J. Clark Jefferson, A.R.S.M.Wh.Sc. Vols. VII and VIII. Chesterfield and Derbyshire Institute of Engineers.

Triger's Method.—This method is applicable to depths of 80 or 100 feet. It consits in sinking a drum-headed cylinder of boiler plates which being filled with compressed air prevents the access of water. By means of an air-trap workmen can enter the cylinder and carry on the work of excavation. It was first employed on the Loire, and in a modified form it is now being used in Victoria, Australia. The pressure of the atmosphere can not exceed four atmospheres

without affecting the workmen, and thus the depth of descent, as in diving, is limited.

Guibal's Method.—This method consists in forcing downwards a cylinder of wood or iron which has a diaphragm across its center from the middle of which a large tube rises upwards. As the cylinder descends, the water rises in the tube, and curbs are inserted on the inside of the sliding cylinder. Obstacles are removed by tools which are passed down the tube.

Kind and Chaudron's Method.—By means of large *trepans* (boring tools) a shaft 4 or 5 feet in diameter is excavated, which is enlarged by other tools to 10 or 15 feet. After this a series of iron cylinders are gradually sunk in the excavation and the hole lined. The details of sinking the lining and rendering it water tight, are important features in this system. It has been successfully employed in several countries. The system necessitates considerable strength in the sides of the shaft.

Since 1878, altogether 16 shafts have been sunk by this method—5 in Belgium, 3 in France, 7 in Germany, and 1 in England.

Poetschs' Method.—On the area where the shaft is to be sunk, a series of holes are put down and lined with copper tubes through which a freezing mixture is caused to circulate until the ground has solidified, after which the excavation can be carried on. The method is not unlike that practiced by gold prospectors in Siberia, who reach the bottom of a river by gradually cutting a pit through the ice and allowing it to thicken beneath. After thickening, the excavation is deepened.

In May 1891 at the Lens Collieries, the site for a shaft 137 feet 9 inches deep was being frozen by a solution of calcium chloride at—12° C being circulated through the lining of 28 bore holes. Altogether to complete the freezing it was expected to take 120 days.

Iron Tubbing.—If tubbing is in complete rings as in the Kind and Chaudron system, it is difficult to put in and difficult to repair. It is therefore usually built up as a series of flanged plates, ¾ inch to 1½ inch thick and from 1 to 3 feet square. Between the joints ⅜ inch to ½ inch of sheathing is inserted. The *wedging curb* at the bottom is in the form of a hollow box. If the flanges of the plates

are on the inside they are convenient for the attachment of the shaft fittings. As iron is liable to decomposition, especially in up-cast shafts, it may be varnished or lined with brick or wood. Should any fractures occur, they are unfortunately hidden by the lining.

Shaft linings, whether of iron, masonry or wood should be stronger below than above. The thickness of the lining varies with the diameter of the shaft and the height of the column of water to be supported.

If R=the radius in meters of the circle inscribed on the exterior of the polygon.

H=the height of the column of water in meters then Pernolet gives the following formula for the thickness of the tubbing.

E=thickness in meters.

$$\text{For wood } E=\frac{H\,R}{1{,}200}$$

For cast iron as used by Kind and Chaudron.

$$E=\frac{H\,R}{20{,}000}$$

For greater safety Chaudron takes.

$$E=\frac{H\,R}{500}+0.020$$

For the thickness of cast iron tubbing Greenwood gives the following formula.

X=required thickness in feet.
P=vertical depth in feet.
D=diameter of pit in feet.

$$X=.03+\frac{P\,D}{50{,}000}$$

The thicknesses actually employed are usually determined from experience.

Cost of Sinking (*Agendas Dunod*).—The average cost of sinking a shaft 4 meters in diameter or 16 meters, square, per running meter is approximately as follows.

1.	Solid ground, no walling.....	350 to	400 francs.
2.	,, ,, with walling.....	500 to	600 ,,

3.	Fissured ground, water in small quantity requiring timbering but not pumping	900 to	1,200	,,
4.	Ground traversed by the Kind-Chaudron system and lined with iron	2,300 to	4,000	,,
5.	Watery ground requiring heavy pumps	2,000 to	24,000	,,
6.	Superficial sands not more than 30m. in thickness, traversed by compressed air	4,000 to	8,000	,,

Cost of Sinking in Coal Measures (*Merivale*).—

Total labour cost of sinking and walling	per fathom	£ 25
Contractor for sinking and small stores	,, ,,	,, 14
Making walling beds each	,, ,,	,, 6
Walling with fire clay lumps	,, ,,	,, 15
Tubbing with cast iron	,, ,,	,, 90
Plank brattice	,, ,,	,, 2.10
Guides of wood	,, ,,	,, 0.15
Guides iron or steel (50 lbs per yard)	,, ,,	,, 2.10
Wire rope guides	,, ,,	,, 1. 5
Total cost of finished pit about	,, ,,	,, 50.00

Cost of walling a pit 14 feet diam. with stone may be from £15 to £ 20 per fathom. Metal tubbing will cost £ 50 to £ 60 per fathom exclusive of fixing which may be £10 per fathom extra. Brick lining may cost ⅓ less than stone.—(See cost of breaking ground p. 98).

Shaft Fittings.—Where shafts approximate to a steeply inclined level, steps may be cut in the floor, or they may be boarded to form a staircase. Near Salzburg men enter the mine by sitting on wooden rails and sliding down the inclined shaft. They return by steps. When sinking, men descend in a kibble or on a cross stick. In parts of Spain they descend by sitting with one leg in a loop, a second loop passing round the chest.

Ladders.—Ladders are chiefly used in metal mines. Men can ascend on stemples. Ordinary ladders consist of two cheek pieces usually rectangular in section say 2″ × 4″, in. by in. set 12 inches to 14 inches apartby a series of staves. The staves are either rectangular or circular in section and thickened in the middle, placed 10 inches apart. Iron staves are trying to the hands and when worn become sharp, but they last much better. An angle of 70° is a good inclination for a ladder. When vertical the center of gravity of a man falls outside the ladder and the strain on the muscles in lifting causes unhealthy fatigue. The foot of each ladder rests on a stage, the upper end passing through a man hole (*sollar*) in an upper stage. Underground ladders do not usually exceed 6 fms.; 2 and 3 fms. lengths are common.

In shafts with the sets 4 ft. apart—ladders may be usually about 16 ft. long—the *sollar* being on every 3rd set.

Chain ladders are often used in sinking. They can be lengthened as the sinking proceeds and they are not readily damaged while blasting.

Ladders may be used to depths of 100 or 200 fms.

Work done on Ladders.—In a mine 200 fms. deep from the bottom of which 300 miners each weighing 150 lbs. ascend daily, the work done in ascending is $200 \times 6 \times 300 \times 150 = 54{,}000{,}000$ foot lbs. and in a year of 300 days the work is $16{,}200 \times 10^6$ foot lbs.

If a days' work is worth 4/- and a man accomplishes 1,000,000 foot lbs. per day the value of the above work would be £3,240 or sufficient to pay for the employment of engine power, and thus enable the miners to go and return from their work without fatigue.

Man Engine (*Fahrkunst*).—These were first employed at Clausthall in 1839. They were independently invented and employed in Cornwall in 1842. Collins says that a man engine was used at the Tresavean Mine, Cornwall, in 1835.

A single man engine consists of a series of rods, like pump rods, which slowly oscillate up and down the shaft. At intervals corresponding to the length of stroke, steps are placed. At the end of each stroke these steps are in front and near to a series of steps fixed on the side of the shaft. In descending, a man steps on to the step on the rod, and descends with it until opposite a fixed step on the side of the shaft. Here he steps off upon the fixed step and waits until the rod has moved upwards to bring the second step on the rods in front of his position on to which he moves and descends to a second fixed step in the shaft.

In a double acting man engine two rods move side by side but in alternate directions. Each of these is provided with steps which come opposite to each other at the end of each stroke.

By moving from a step on one rod to that on the other at the end of each stroke a miner may ascend or descend.

Motion is given to the rods by a crank working a rocking beam. By this means, although the crank has a continuous motion, at the

dead points of revolution the motion of the rod is reduced to zero, and the miner has time to move from step to step.

The balancing and the regularity of motion of two rods is secured by 1. Coupling the rocking beams together. 2. By a rack on each rod working in a pinion placed between them. 3. By connecting the two rods by a chain passing over a pulley fixed in the shaft. 4. By plungers attached to the rods, each working in a chamber containing water, the two chambers being connected.

The speed may be regulated by the rod raising a plunger which as it rises sucks water in at a valve. As it descends, the water escapes through an opening the size of which may be regulated.

For safety in case of accident *catch pieces* are placed at intervals on the back of the rods which overhang or embrace bearers crossing the shaft. If a breakage occurs the rods may either fall or in consequence of the balance bobs fly upwards. The catch pieces coming in contact with the bearers regulate the length of this motion.

Length of stroke 6–12 feet. The rods are 7 or 8 inches square. Sometimes they are made of angle iron or iron rope. The weight of the rods with all connections is about 25 cwt. per fathom and the cost of a man engine with driving engine complete for 200 fms. is about £2,000 to £2,500 (*Collins*). For farther details see "Mining Statistics West of the Rocky Mountains." *Raymond and Blake.*

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LEVELS.

Variety of Levels.—Adit, Sough (Derbyshire) Gutter (Staffordshire). Shallow adits, deep adits. Drift, headway, cross cuts, cut outs, branches, tunnels, roads, gates, rollyways, jig brow, durk drifts. The end of a level is called the forebreast or end.

Dimensions of Levels.—Levels for exploring or *costeaning* may be 2½ by 4 feet. Main levels are usually from 3 by 5 feet to 4½ by 7 feet. In metal mines they are usually the width of the lode, while in coal mines they are usually the height of the seam. The width greatly depends upon the strength of the roof. A double road for two lines of rails may be 4 or 5 yards broad.

In England a drift (6 feet × 9 feet) thought coal and soft materials costs about 30/- per fathom.—(see p. 99.)

Adit or drainage levels may be 8 to 12 feet high and 5 to 8 feet broad. In soft ground the gradient may be 1 in 200 (3/16 in. per yard) and in hard ground 1 in 600 and even 1 in 1,000.

The County adit in Cornwall drains 30 square miles. It is 40 m. long and 30 to 90 fms. deep.

The Beaumont adit is 6,567 yards long and 70 fms. deep, gradient 1 in 660.

The Ernst August adit, Clausthal, is 18 m. long (25,956 meters). Greatest depth 222 fms. Sectional area 10 by 6 feet. Gradient 1 in 2,000. It took 13 years to make and cost £85,500.

The Sutro Tunnel 4 miles long (Comstock Lode) is 10 by 12 feet square and 2,000 feet deep. It was completed by 9 distinct holings.

To drive a long level quickly, work by means of shafts from several points, increase the number of shifts, use machines, employ picked men, and drive a narrow gallery ahead.

Position of Levels,—Usually in a lode but if outside you get an uninterrupted road. If partly on a foot wall you obtain a good gutter for drainage but a bad roof.

In lodes with slight dip, the levels are close when the distance between them is measured vertically, if far apart, the expense of putting in pumps may be great. In determining the distance apart, consider the cost of driving, the cost of underground transport, which is usually cheaper if there are many levels, the distribution of the minerals, the method of working &c. In Cornwall levels were usually 8 fms. apart, now they are 10 and 15 fms. apart. In Saxony 20 to 40 fms. apart.

Timbering and Walling.—As to whether an excavation shall be timbered or walled, depends upon the material and workmen in the district, the length of time that the excavation is to remain open, &c. Timber is put up more quickly, is cheaper, and occupies less space than masonry. It accomodates itself to small movements. Masonry is stronger and more durable than timberwork, but it can only be put up slowly; also it may crack. Levels are sometimes supported by iron. Where old rails can be employed, it is cheap and can be erected quickly.

Varieties of Wood.—Leaf wood has a compact core but is soft outside. Pine wood is soft throughout and contains turpentine. Pine also has sap wood.

English Oak is durable under all circumstances. It may last 25 years or more. It is strong but occurs in short lengths. *Beech* and *Birch* are strong but liable to *red rot*, and decompose in a stagnant atmosphere. *Acacia*, resists decomposition; tough and flexible. *Ash*, may be bent; used for handles, rungs of ladders. *Elm*, used for frames and curved work.

Hard wood like oak and elm used for cribs, costs in England 3/– to 4/– per cubic foot.

Pine decays in a bad atmosphere, but is light and cheap and therefore much used. In England, Scotch Fir (*Pinus Sylvestris*) and Spruce (*Abies communis*) or red and white wood, is imported from Norway and Sweden. For pump spears, headstocks, pitch pine is imported from America. In Cornwall, Norwegian pine delivered at the mine costs about 1/– per cubic foot. American pitch pine used for pump spears costs 1/9 to 2/6 per foot. *Larch* is similar to pine but containing more rosin is not so liable to rot. *Fir* is more brittle than Pine and Larch, and more suitable for timbering in water.

Wood is best cut in winter and barked to dry. It should be stored in sheds where it will be dry and be open to a free circulation of air. In coal mines timber is often destroyed by crushing. By *dry rot*, timber is converted into a white powder. By *damp rot*, the timber becomes slimy. It usually occurs where there is variation in moisture. Under pressure, timber may be destroyed in a few weeks, but under good condition oak may last 25 years, and pine 6 or 8 years. In old workings, timber which has been constantly wet, has lasted from 100 to 300 years. In some of the Hartz mines the shaft timber is purposely kept wet.

In English collieries the cost of timbering is from 4d. to 9d. per ton of coal worked.

Timber may be preserved by charring, covering with tar, lime or cement. By removing the sap by water or steam and then filling up the pores with salt, zinc sulphate, copper sulphate, tin, zinc or mercury chloride. By treating first with phosphate of soda, and then with barium chloride; barium phosphate, which is a solid stoney substance, is precipitated in the pores of the wood, while the soluble sodium chloride, which is simultaneously formed, may be washed away.

Timber saturated with creosote has a strong smell and therefore objectionable when used underground.

The square section of a piece of timber which can be cut from a given log is approximately four-fifths of a quarter of the square of the mean circumference.

Timbering Levels.—In coal mines many *props* or *posts* with a *lid* or *cap* above and *sill* beneath are employed. They may be withdrawn by a hooked lever and chain called a *dog*. If the upper ends of legs are hollowed they are apt to split. Legs may be prevented from being forced inwards by wedging, nailing, notching, or a strut. Legs and their *caps*, *head pieces* or *collars*, may be united by scarfing, the nature of the joint being regulated by the relative amounts and direction of pressure.

Stemples or stull pieces may have to support a hanging wall and a load of *attle* or *deads*. One end rests in a *stemple-notch* or *hitch*. The stemple is usually so placed, that any movement of the hanging wall tends to wedge it tighter, by bringing it into a direction perpendicular to the two walls. The *rise* or difference from the perpendicular

in the direction of a stemple may be from 10° to 15°. A similar tendency to wedge may be obtained by properly shaping the ends of a stemple. Similar principles are followed in inclined coal seams. Wedges or *lids* may be used at the end of *stemples*, corresponding to *cap pieces* used with props.

Box, nog or chock timbering, is used for great pressures or high roofs.

A simple form of roof timbering, consists of a series of stemples covered with lagging. In wide lodes, stemples may be supported by struts from the hanging wall. In flat lodes, the strut rests on the foot wall. Stemples may be supported by struts from both walls. Rafter timbering, is where two stemples rise from the walls to meet in the middle of the lode where they but against a plank.

In stratified deposits, levels are usually supported by two leg pieces and a cap. If the levels are wide, a central prop or strut from the legs may be required to support the cap. Above the caps, *lagging* may be employed and the frames or door sets, consisting of leg pieces and caps, may require backing behind, and *stringing pieces* to keep them apart. Stringing planks, held in position by struts or stemples, are employed to support the sides or walls of a lode with a soft floor. Sole pieces or sills are placed beneath the legs or stanchions.

In salt mines the floor is sometimes timbered with an inverted arch of wooden blocks. A level may be completely arched in this manner.

Driving in Soft Grouud.—*Spilling*, is timbering which is carried on simultaneously or partially in advance of the work of excavation, in running or quick ground.

The spills are planks about 2 inches thick, 4 inches wide and 6 or 7 feet long. These are sharpened by slightly beveling one end so that when they are driven into the soft ground horizontally they rise or diverge. The method consists in driving spills into the end of a level and then excavating. The spills are supported as the excavation proceeds. If the ground is very quick, *breast* or *closing boards* will be required to support the face of a level. This method has some resemblance in principle to that which, in 1823, enabled Brunnel to drive the Thames tunnel, the sectional area of which was 38 feet by 22 feet 6 inches. He employed a shield of iron made up of 12 frames

with 3 cells in each. The shield was forced forwards by hydraulic rams, and brick work inserted in the space thus gained. To relieve the pressure behind, the shield the cells could be opened.

Mr. Victor Simon, a Belgian engineer, devised a plan for penetrating soft ground which has succeeded where ordinary spilling has failed. This consists in covering the floor, roof and face of a level with a mass of conical wedges each about 3 feet long. As these are driven forwards, the space which is opened is filled with solid door sets.

Archwork and Walling.—The thickness of an arch in feet $= .0694 \times$ radius in feet $+ 1$ foot. It should not be less than 9 inches.

The height of crown $= 5$ inches to 6 per yard of span. Under pressure, 8 inches per yard. For inverted arches 3 inches per yard.

An arch 2 feet wide and 11 inches thick will hold 45 feet of attle. The pressure is the weight of the attle less its own friction.

In an inclined lode, arches to carry attle are placed so that their springing line is at right angles to the dip.

Side walls have a batter of 1 inch to 1¾ inch per yard. For watertight walling, the joints are increased from ¼ inch to ½ inch. Walls are destroyed by splitting, buckling, shearing and overturning. For a wall not to be overturned, the line of resistance, which for a rectangular wall is an hyperbola, must fall inside the foot of the wall. Crushing strength in lbs. per square inch.

Brick	500— 1,000 lbs.
Limestone.....	4,000— 5,000 „
Granite	5,000—10,000 „

Dry walling only, resists crushing. In English bond, a course consists entirely of headers or of stretchers, while in Flemish bond, each course consists of headers and stretchers. Open spaces between masonry and the roof or sides of levels must be carefully packed. Sand forms a good packing.

Cost of Arching (*Collins*).—For a drift 8 feet × 7 feet one brick thick, with lime mortar and well rammed behind with clay.

	£	s.	d.
1,500 bricks at 20 shilling per thousand	1.	10.	0
Lime 2 cwt. at 9 d.		1.	6
Sand 6 cwt. at 2 d.		1.	0
Labour of laying and pugging behind		10.	0
	2.	2.	6

If there is an inverted arch below the cost is ⅓ more. If half a brick is sufficient the cost is ⅜ less.

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HAULAGE.

General Notes.—In mines with irregular roads it is necessary to transport materials in bags, basket or trays. In working the *churns* of iron stone in the Forest of Dean. boys carry from ½ to ¾ cwt. in a tray called a *billy*. To carry coal along a working face to a gob road, *sleds* are often used. Originally the coal was placed in large baskets called *corves*, which were placed on the sled and carried to the bottom of the shaft.

Narrow wheelbarrows with a low center of gravity are sometimes used underground. In Cornwall a wooden barrow without legs, costing from 8/6 to 12/6 carries from 1 cwt. to 1¼ cwt. The *hund* is a four wheeled waggon used in Germany. The front pair of wheels are placed almost under the center of gravity. When it is moved it is slightly tipped so that it runs on two wheels, while when standing it rests on four.

Trolleys are four wheeled flat trucks without sides. In 1775 Mr. Carr of Sheffield used iron tram plates.

Nicholas Wood, who experimented on haulage, says that on a good road on the surface, a horse will draw 133 tons 1 mile per day. Underground with wheels 1 foot in diameter it may draw 30–50 tons, but if the wheels are less, it may only draw 15 tons.

Kibbles or Barrels.—Of wood or ¼″ sheet iron; 22″ at top, 34″ in the middle, and 15″ below. Skips or boxes which run on guide rollers are about 2′ square and 4 or 5′ long. They carry 6 to 10 cwt.

Kibbles are usually suspended on trunions which pass through holes in the handle, the point of suspension being below the center of gravity of the kibble.

Tubs.—Tubs ought to be light and strong. The wheels may be, 1. Fixed on the axle, as on ordinary railroads. 2. The axle may be fixed and the wheels revolve on the axle. 3. One wheel loose and one fixed to the axle. 4. There may be as many axles as wheels.

In the first system when passing round a curve, inasmuch as the two wheels on the same axle do not pass over the same distance, one of them must be rubbing while the other is rolling. In running round a curve in consequence of the inertia of the tub the flange of the outer

wheel grinds against the inner side of the outer rail. Wheels which are slightly coned partially obviate this. In passing round a sharp curve, two axles of a tub do not tend to remain parallel. To allow for this tendency to deviate from parallelism, play of say ¾ inch is given to the wheels. If a tub is turned round on successive journeys along the same curves, the wearing of wheels and bearings such as those just mentioned may be equalized. The flanges of wheels are slightly rounded. Such a form offers resistance to derailment and a tub is easily placed on its track. Wheels are usually 7½ to 15 inches in diameter and axles 1¼ inch to 1½ inch.

The height of tubs may depend on the thickness of the seam. Small tubs carry 3 to 4 cwts. Ordinary tubs carry 6 to 11 cwts. In S. Wales there are tubs carrying 1 to 1½ tons. Large tubs are difficult to handle. Percentage of dead load to useful load is from .25 to .50.

A waggon 30″×42″×20″ holds about 1 ton of iron ore.

Tubs carry about 50 lbs. of unscreened coal per cubic foot of contents.

Ordinary tubs are usually constructed of hard wood, like elm or oak; ¾ to 1 inch thick, bound with angle iron. The rectangular boxlike form is the commonest. Prismatic forms are also used. Iron tubs are lighter for a given capacity but they are difficult to repair. Rectangular forms slightly bellied, are considered good designs. Length 3′.7″, breadth 2′.6′, depth 1′.10″. The axles turn on steel bearings. On each side there is a fixed and a loose wheel. Wooden tubs at Pas de Calais weighed 110 k. and carried 400 k.

At Blanzy, loose wheels on loose axles have been used. The journal box is oval and the play thus afforded diminishes the chance of derailment. Well greased washers keep the wheels apart.

At the Pleasley Colliery, tubs carrying 13½ cwt. with 13½ inch steel wheels, 2 feet gauge, wooden sides 7 sheet iron ends, weigh 7 cwts.

By elbowing the axles, the bottom of the tub comes below the axles. In this way the height is reduced, while a large capacity and an increased stability are obtained.

Steel tubs, as for instance those made by Hudson & Co., carrying 11 cwts. of coal and costing £4.10.0 are durable and satisfactory.

At coal mines, on reaching the bank a tub is run into a *tippler* or *tumbler*, and its contents discharged upon the screens.

Resistance to Traction or Rolling.—The resistance to traction may be estimated,—

1. By determining the force required to keep a tub in motion. This may be done by attaching an apparatus like a strong spring balance to the line which pulls the tub.

2. By finding the least inclination which can keep the tubs running. The tangent of this slope is the coefficient of resistance to traction.

3. By observing the distance up an incline which a tub will run after having descended another incline.

4. By noting the time taken by a tub in running down a slope of given length and given inclination (*Merivale*).

Let L = length of the incline in feet.
H = height „ „ „ „ „
T = time taken to descend in seconds.
W = weight of tub in lbs.
R = friction in lbs.
a = angle of inclination of incline.
g = gravity = 32.

$$R = W\left(\sin a - \frac{2L}{gT^2}\right)\cos a.$$

This friction usually varies between $\frac{1}{10}$ and $\frac{1}{100}$ of the weight of the loaded tub. On ordinary railroads at low speeds it is about 8 lbs. per ton. With tubs it is from 2 to 4 lbs. per cwt.

The resistance to rolling on a level may also be expressed.

Let W = the load in a tub.
W_1 = the weight of the waggon or tub and its frame.
W_2 = the weight of the wheels.
$W_1 + W_2$ = the weight of an empty waggon.
f = the coefficient of friction of the axle and its bearings. Say .018 to .035.
F = coefficient of rolling friction. Say .001.
r = the radius of the axle.
R = the radius of the rubbing portion of the wheels.
S = the total resistance to rolling on a level.

Then $S = F(W_1 + W + W_2) + \frac{r}{R} f(W + W_1)$.

The coefficients given are for Railroads. *See Molesworth.*

Resistance to be overcome, or force of traction on a slope of inclination i.—

Ascending.

$$S_1 = \frac{fr}{R}(W + W_1) + (W + W_1 + W_2) \sin i.$$

Descending.

$$S_2 = \frac{fr}{R}(W + W_1) - (W + W_1 + W_2) \sin i.$$

This and the next two formula are given as in Callon. On a badly laid tramway rolling friction may be considerable and should be introduced as in the last formula.

To reduce the force of traction, make f small by polished and lubricated surfaces, make the axles small, increase the radius of the wheels.

In reducing the value of r, the strength of the axle has to be considered, while when increasing R the height of the tubs must not be overlooked. In practice we may have.

$f = .1.$

$\frac{r}{R} = .1$ { For wheels .20m. to .25m. in diam. this may be increased, while for wheels of .50 m. diam it may be less, say $1/12$.

$W_1 + W_2 = .4\ W.$

$W_1 = 2.5\ W_2 = .29\ W.$

$W_2 = .11\ W.$

$S = .0129\ W$ on a level for a full waggon.

$S = .0029\ W$ for an empty waggon.

For a full waggon the resistance is therefore about four times more than for an empty waggon.

Slope of equal Resistance.—This is a slope on which the work done in taking the full waggons towards the shaft, equals the work done in returning with the empty waggons. This slope i is given by the following formula.

$$\text{Sin } i = \frac{fr}{R} \cdot \frac{W}{W + 2(W_1 + W_2)}$$

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The value for sin i in practice is about .0055 and the slope is 5 or 6 millimeters per meter or 1 in 200. André gives 1 in 111.

On such a slope the force of traction or $S_1 = .0051\ W$, or 5 thousandths of the weight transported.

Slope of Equilibrium.—A slope of equilibrium is one where the full waggons when once started will continue in motion by themselves. In such a case the trammer rides behind the full waggon to the end of its journey, his only work being to push the empty waggons back to the starting point. Such slopes are used on long well constructed roads.

Such a slope is given by the following formula.

$$\text{Sin } i = \frac{fr}{R} \cdot \frac{W + W_1}{W + W_1 + W_2}$$

In practice Sin i may equal .0092, or 10 millimeters per meter or 1 in 100. André gives 1 in 61, but it might be 1 in 88.

To bring back the empties on such a slope the force of traction $F_1 = .0066\ W$ or 6 to 7 thousandths of the weight moved.—(*See Callon.*)

Roads.—The narrower a road the less resistance is there on the curves but the tubs are more unstable. Rails are often flat iron bars 1½″ by ½″ to 2¾″ by ¾″. These are fixed in sleepers 2′ to 2′.3″ apart by wedges driven on the inside. The ends of two rails should terminate at a notch. Bridge rails, *T* shaped and double headed rails are used on main roadways. They weigh from 10 to 20 lbs per yard of length. The ordinary rail now used is flat bottomed and weighs 12–15 lbs. per yard. The gauge 12″–24″. Minimum radius of curves 6–10 feet.

Bridge rails weighing 14 lbs. per yard spiked with 2″ nails to sleepers 7″ wide 2″ thick, cost and about 3/– to 4/6 per yard. The gauge may be 36″.

Elevation of Rails at a Curve (*Molesworth*)—

Let h = elevation of outer rail in inches.
W = breadth of line in feet.
V = the velocity in miles per hour.
R = radius of curve in feet.

Then $$h = \frac{V^2}{1.25\ R.}$$

The section and weight of rails vary with the load they carry, and the gauge.

At junctions where a train of tubs has to pass, movable points are used. For single tubs these are not required. At rectangular crossings where tubs may be turned, iron plates and guide rails are employed. With inclinations of more than 1 in 30 self acting planes where the full tubs descending pull up the empty ones are used. For very steep inclines the tubs are carried down upon trolleys with a level floor. On self acting planes, brakes are required. These may consist of a segment of wood or an iron band. To prevent a rope slipping it may be carried several times round a single pulley or back and forth round two pulleys. A pulley with a conical groove lined with wooden plugs grips a rope. Fowler's clip pulley answers a similar purpose.

To lessen the chances of accidents from runaway tubs on self acting inclines, at the top of the incline there should be *stops* to hold the tubs, and at the bottom *seizers* to hold the rope or chain.

To avoid friction on a self-acting plane, the sag of the rope is carried on sheaves placed between the rails.

Self-acting Inclines (*Merivale*).

Let L = length of incline in feet.
H = height of incline in feet.
a = angle of incline
F = weight of full set in lbs.
E = weight of empty set in lbs.
T = time running in seconds.
g = gravity = 32.
R = weight of rope in lbs.
S = weight of rollers and sheave in lbs.
m = coefficient of friction of tubs on level road.
m' = coefficient of friction of rollers and sheave = about .03 on average.

W = weight in lbs. of the whole mass in motion.
P = force in lbs. moving the sets.

Then $P = F \sin a - \{mF \cos a + mE \cos a + m'S + E \sin a + R \sin a\}$

$= F \sin a - \{m \cos a(F+E) + m'S + \sin a(E+R)\}$

but $\sin a = \frac{H}{L}$ and $\cos a$ practically $= 1$.

$$\therefore \quad P = \frac{FH}{L} - \left\{m(F+E) + m'S + \frac{H}{L}(E+R)\right\}.$$

$$T = \sqrt{\frac{2LW}{g\left\{\frac{FH}{L} - m(F+E) + m'S + \frac{H}{L}(E+R)\right\}}}$$

$$\frac{H}{L} = \frac{m(F+E) + m'S + \frac{W2L}{gT^2}}{F-(E+R)}$$

Comparison of Work done by Men and Horses.—From experiments made years ago at the mine Vicoigne, it would appear that the gain in employing horses underground to do the same work as men or boys may be expressed as follows.

If the gain in money on carrying 100 m. be x. then the gain.

For	200m.	=	$2.7x$.
,,	300	=	$3.4x$.
,,	400	=	$5.4x$.
,,	500	=	$6.2x$.
,,	600	=	$6.1x$.
,,	700	=	$6.7x$.
,,	800	=	$7.2x$.
,,	900	=	$7.4x$.
,,	1000	=	$8.5x$.
,,	1200	=	$10.5x$.

The useful effect is greatly influenced by the nature of the road. Burat says that at Charleroi with a straight road and a slope of .005m. per meter, a young woman may perform the work of 1,500 kilos per kilometer. At Mons the same work is difficult for a man.

Haulage by Steam Power.—1. *Tail Rope System.* A train of 10 or 20 tubs is drawn by a rope coiling on a drum, at the same time a smaller rope (*tail rope*) is uncoiled from a second drum running freely on a shaft. This latter rope passes over a system of pulleys, and is attached to the end of the train, so that when the train has reached its destination near the hauling engine, the second rope has been uncoiled for a length equal to double that which the tubs have travelled and is therefore ready on reversing the engine and throwing its drum into gear with the shaft, to draw a train of empty tubs back to the point from which the full tubs started.

2. *Endless Rope* or *Chain Systems.* In these systems a chain or rope passes round two large pulleys, one situated at each end of the line along which the tubs are to travel. One of these pulleys is driven continuously and slowly in one direction so that part of the rope or chain is continuously travelling in one direction over the center of one line of rails and the other in an opposite direction over the center of a another but parallel line of rails. The full tubs are fixed at intervals on to the rope or chain travelling over one set of rails, while the empty tubs are similarly fixed on the rope or chain over the other line of rails.

In the 1st system two drums are each provided with brakes and can be separately thrown in or out of gear with the shaft. When hauling full tubs the tail rope drum is out of gear. When empty tubs are returning the tail rope drum is in gear and the main rope drum is out of gear. The tail rope is carried along or down a level on a system of pulleys placed in an angle of the roof. Up the center of the line of rails a series of sheaves are placed to lessen the friction occasioned by the main or tail ropes sagging on the ground. Here a single line of rails is only required but one engine may drive four drums to be used on two different levels. Time is lost in the return journey, but this is compensated for by high speeds varying from 12–16 feet per second (10–15 miles per hour). Strong ropes are required.

3. *Main Rope System.*—With inclination of more than 1 in 28 the tail rope may be dispensed with, as empty tubs by gravity alone will drag out the main rope as they run back towards the starting point. To avoid the running away of tubs behind the last in a train, a *trigger* or *devil* is placed, which so long as the tubs are ascending

trails on the ground. A *bridle chain* will hold a train of tubs together.

With the no menclature of p. 123.

$$P = E \sin a - (m \cos a\, E + m'S + m'R).$$

and we get

$$P = \frac{EH}{L} - \{mE + m'S + m'R\}.$$

$$T = \sqrt{\frac{2LW}{g\left\{\frac{EH}{L} - (mE + m'S + m'R)\right\}}}$$

$$\frac{H}{L} = \frac{mE + m'S + m'R + \frac{W2L}{gT^2}}{E} \qquad (\textit{Merivale}).$$

In the endless rope or chain system to prevent a rope from slipping, it either passes several times round a pulley or a special pulley (Fowler's clip pulley or a V groved pully fitted with plugs of wood) may be employed. Fowler's clip pulleys are usually employed to transmit power to a wire rope used for haulage, or to turn a shaft at a distance. These pulleys are at either end sufficiently high to clear the tubs, but the chain by its sag reaches them at a short distance, where it catches in forks fixed to the top of each tub. Sometimes the rope or chain runs beneath the tubs and these are attached to the rope by clips (Hanson's Clip. Humble's Clip.)

The speed is usually about 6 feet per second (4 miles per hour). At the upper end of the line the tubs run down an incline and catch the chain automatically. By an incline at the lower end they disengage themselves automatically. A similar arrangement enables a tub and the chain to pass round a curve, at which place the chain is raised by a guide pulley. As the curve is approached the tub is disengaged by the chain being raised, and it runs round a slope and again engages itself with the chain after passing the curve. The tubs are run on to the line at intervals of 20 or 30 yards. After a tub has reached a certain point, it may be caused to give an automatic signal indicating that another tub may be started. The weight of the chain adds to the frictional resistances on the line.

With a given velocity, more tubs can be conveyed in a given time than by other systems. On an incline, the weight of the full tubs assists in raising the empty tubs.

The endless chain system is suitable for straight roads which may be undulatory. The tail rope system may be employed on roads which are crooked or branched but not undulatory.

In considering the relative merits of these systems we have to consider. 1. First cost of waggon way. 2. First cost of plant (engines, rails, &c.) 3. Cost of conveying coal per ton per mile. 4. Cost of maintenance (ropes, chain and plant generally). 5. H. P. required, &c.

COST OF HAULAGE (*André*).

The Following Table was drawn up by a Commission appointed by the North of England Institute of Engineers.

System.	Coals per day, tons.	Average distance in yards.	Average gradient for full tubs.	Horse power exerted.	Cost of labour per day.	Cost.		Per Ton.		Per Mile.		First Cost.		
						Ropes and Chain.	Maintaining plant.	Coals.	Labour.	Total.	First cost of waggon way.	First cost of engines, boilers, &c.	Cost per mile of waggon way.
			1 in.		£ s. d.	d.	d.	d.	d.	d	£	£	£
Tail Rope.....	476	2,133	213	112.36	1.07.7	.276	.462	.558	.583	1.879	3,398	1,105	723
						15%	24%	30%	31%				
Endless Chain.....	451	1,389	59	20.20	13.8	.083	.468	.256	.572	1.379	1,755	276	941
						6%	34%	19%	41%				
No. 1 Endless Rope.	384	922	48	62.38	17.9	.263	.541	.237	1.020	2.061	1,500	553	820
						13%	26%	12%	49%				
No. 2 Endless Rope.	443	849	36	29.01	1.10.1	.252	.726	.323	1.692	2.993	2,322	973	893
						9%	24%	11%	56%				

In the tail rope system the total cost per ton per mile varies from 1.948 to 2.172d. In endless chain from .551 to 1.8d. and in endless rope from 2.061d.

Cost of Transport per Ton per Mile (*Merivale*).—

¾ d. Ordinary railroad with locomotive.
6 d. Ordinary road with horses, excluding maintenance of road.
¼ d.(?)Canal with horses, excluding maintenance of canal. Canals with steam-tug as low as $^1/_{100}$ d. exclusive of maintenance of canal.
1¼ d. Level railroad with horses.
2½ d. Underground level rolly way with horses.
2 d. Ordinary underground rolly, rope or chain haulage.

Electrical Haulage.—In the arrangements for electrical haulage a motor by gearing drives a drum. In a plant put up by Messrs. Crompton as the Abercanaid Colliery S. Wales to replace 27 horses, the motor is built to run at 800 revolutions and takes 80 amperes at 850 volts. The Dynamo will give 180 amperes at 500 volts running at 550 revolutions. The cable is 3,200 yards long and composed of 37 strands of No. 14 high conductivity copper. Its resistance is 3,192 ohms. and there is a loss of potential of 51 volts or 10 per cent.

Underground Locomotives.—These have been used in certain coal mines in Pennsylvania. Weight 1,100 lbs. wheels 2 feet diam: gauge 3 feet 6 inches. Length 12 feet. Height 6 feet. The first cost and subsequent working expenses are less than when mules have been used.

Underground Canals.—Under favourable circumstances, as at certain mines in Silesia, at Clausthal, in Lancashire and Flintshire, levels partly filled with water are used as underground canals.

Wire Tramways.—For the conveyance of material upon the surface, a load is carried upon a continuously moving wire rope, or by haulage along a rope which is fixed. The cost of one of the former lines capable of transporting 100 tons a day is about £400 per mile, while maintenance and cost of transporting is about two pence per ton per mile.

Hallidie's wire tramway in principle of working is like the endless rope system. The recepticles for material are suspended on the

rope which works round clip pulleys at the ends of the line. Cost $6,000 to $8,000 per mile. Runs 200 feet per minute and with buckets 100 feet apart each carrying 100 lbs. delivers 6 ton, per hour.

A single endless cable near the Ashio Mines, Japan is as follows :—

Length 3760 metres, Supports 17. Longest spans 550, 423 and 350 metres. Maximum load per bucket 200 lbs. Spacing 100 metres. Speed 50 metres per minute. H. P. about 8 or 9. Roe and Bedlington's Automatic clip saddles are used which prevent slip on steep gradients and also during wet weather. Also patent balanced sheaves are used. These latter increase the angle taken by the cable at the trestles, adjust themselves to take an equal share of cable pressure, and prevent local pressure or set.

With the same system in which there is equal wear over the length of the cable, loads of 600 lbs. or more may be carried.

In some systems stationary ropes stretched at the end are used as tracks. On an incline the empties may be pulled up by the full buckets.

WINDING.

Winding Engines.—Some winding engines lift 1,000 to 1,600 tons of coal per day. The cost being about $\frac{3}{4}$ d. per ton per 100 fathoms. At some mines however from 2,000–3,000 tons of coal are raised per day. Double cylinder, high pressure, condensing or non-condensing engines are usually employed. For small loads single cylinder engines with gearing are used. Vertical engines occupy a smaller space and can be placed nearer a shaft than horizontal engines. They require firm supports, and the forces applied to the crank in the upward and downward stroke differ by twice the weight of the piston and its rod. Horizontal engines can be readily inspected, a solid foundation is easily obtained, but the wear on the two sides of the piston, &c. is unequal. These are the most common type.

With compound condensing engines at water works, a consumption of 1.9 lbs. coal gives one effective H. P. and 1.6 lbs. gives one indicated H. P. With small coal at collieries, pumping engines do fairly well if they only burn 3¼ lbs. per I. H. P.

Duty of Engines.—In the best marine engines one I. H. P. per hour may be obtained by the consumption of 1.5 lbs. of coal. In others engines 7 to 8 lbs. may be used. In Cornwall 125 million lbs. of water has been raised by burning one bushel (84 lbs.) of coal. A duty of 65 to 75 millions is common.

Horse Power required at a given Shaft.—

Let H=depth in feet.

T=number of tons raised per day of n hours.

F=the theoretical horse power required.

Then $$F=\frac{H\times T\times 2{,}240}{n\times 60\times 33{,}000}$$

The *effective* horse power or the power actually required to overcome frictional resistances, inertia &c. will be about three times that which is calculated.

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The Resistance due to Inertia.—

Let R = resistance in lbs.
W = weight of load in lbs.
V = maximum velocity in feet per second.
g = force of gravity = 32 feet.
T = time in seconds to acquire the velocity.

Then $$R = \frac{WV}{gT}.$$

Experiment has shewn that when a cage is suddenly lifted the strain on the rope is about double that of the load.

Dimensions of Engines.—

Let H = depth of shaft in yds.
w_1 = weight of cage.
w_2 = weight of tubs.
w_3 = coal raised at each journey.
w_4 = weight of rope unbalanced in shaft.

$\therefore$ $w_1 + w_2 + w_3 + w_4 = W_1$ or weight on one rope and $w_1 + w$ $= W_2$ or weight on other rope.

Statical load at starting $= w_3 + w_4 = W_3$.

If the circumferences of the drum in feet $= C$, the stroke of the engine in feet $= L$, the mean steam presure in lbs. per sq. in $= P$. and the area of the piston in sq. in. $=$ A.

Then $$A \times P \times 2L = C \times W_3$$

or $$A = \frac{C \times W_3}{P \times 2L}.$$

To overcome the friction and the inertia at starting, A may be increased about three times, and from this area, the diameter D of the cylinder calculated.

To overcome dead points, and in case one engine might break down, two engines with cylinders of diameter D are usually employed.

The engine power actually employed is therefore about three times that which is calculated. Ordinary winding engines have a stroke of 4 to 6 feet and a diameter of 24 to 34 inches.

Indicated Horse Power.—

A=area of piston in square inches.
P=average presure of steam in lbs. per square inch in the cylinder.
L=length of stroke in feet.
N=number of revolutions per minute, then,

$$\text{Indicated horse power} = \frac{2\,PLAN}{33{,}000}.$$

Nominal Horse Power.—For scientific calculations this term is meaningless. It is used by makers of small high pressure engines who allow 10 circular inches of piston area per nominal H. P.

Quantity of Material Raised.—To determine the quantity of material which may be lifted per hour, calculate the number of revolutions made by the winding drum each journey, and from that calculate the distance travelled by the piston. Knowing the average piston speed, which may be from 300 to 600 feet per minute, the time taken per journey is readily determined. The time taken for banking may be from 30 to 60 seconds and this added to the time taken for winding gives the time occupied in raising and discharging a given load. With this as a unit, the number of tons discharged per hour may be readily calculated.

The maximum speed of a cage in a shaft may be 50 feet per second.

Boilers.—The boilers usually employed are the Lancashire boiler with two flues or the Cornish with one flue. According to Molesworth for each *nominal* H. P. a boiler requires,

1. Cubic foot of water per hour.
1. Square yard of heating surface.
1. Square foot of fire grate surface.
1. Cubic yard of capacity.
28. Square inches flue area; 18 inches over bridge.

For Lancashire boilers an approximate rule is;

$$\frac{\text{Length in feet} \times \text{Diameter in feet}}{6} = \textit{Nominal} \text{ horse power.}$$

Table of Dimensions of Cylindrical Boilers.

(*Molesworth*).

Nominal H. P.	Length of Boiler.		Diameter of Boiler.		Diameter of Flues.		Number of Flues.
	ft.	in.	ft.	in.	ft.	in.	None.
1	6.	0	1.	9	...	...	...
2	7.	6	2.	0	...	...	...
3	9.	0	2.	6	...	...	...
4	10.	0	2.	9	...	...	...
5	11.	0	3.	0	...	...	...
10	16.	0	4.	6	2.	0	1
15	18.	0	5.	3	2.	4	1
20	20.	0	6.	0	1.	9	2
25	25.	0	6.	0	1.	9	2
30	28.	0	6.	6	2.	0	2
35	30.	0	7.	0	2.	3	2

For more than 35 H. P. two or more boilers are required. Extra boilers are required to avoid stopping the engines while repairing or cleaning.

The student's attention is here called to the meaning of the term *nominal* horse power.

Head Gear. Poppet Heads.—The essential parts of a pit head frame are the *legs* (vertical supports) and the *spurs* (inclined supports).

The height varies from 30 to 60 feet. Wooden legs (American pine) may be 14″ × 18″ square. These and the spurs are secured in iron sockets. The *spurs* or *back stays* are so situated that the direction of the resultant of the forces applied to the pulley,—one by the rope descending the shaft and the other by the rope leading to the drum,—falls within the base of the frame.

Wooden frames may last 30 or 40 years, but there is the danger of their being destroyed by fire.

An iron frame 80 feet high to take a load of 2½ tons should not weigh more than 10 tons (*Collins*).

Formulæ relating to the strength of long and short columns may be referred to in Molesworth's Pocket Book.

When two pulleys are used side by side the ropes lead to different sides of the drum and therefore the angles made by the ropes round the pulleys are different. This may be avoided by placing the pulleys behind or above each other. The pulley frame ought to be adjustable, so that in case of shrinkage of the frame the line of the rope descending the shaft may be made central.

As wire ropes are good electrical conductors the head-gear might be provided with lightning conductors.

Pulleys.—Pulleys are from 6 to 16 feet in diameter. At Kiveton Park Collieries, Nr Sheffield, the pulleys are 20 feet. diameter. The spokes are of wrought iron and the rim and bosses cast. The shape of the groove varies with the shape of the rope.

Let F=force in lbs. applied at rim of pulley to overcome friction of axle.
W=weight of pulley in lbs.
D=diameter of pulley in inches.
d= „ „ axle in inches.
m=coefficient of friction (say .07).

Then $$F=\frac{Wmd}{D}\cdot\cdot \qquad (Collins.)$$

A rope 1 inch circum : requires 10 foot pulley.
„ „ 1¼ „ „ „ 10½ „ „
„ „ 1½ „ „ „ 11 „ „
„ „ 1¾ „ „ „ 11½ „ „
&c., &c., &c.,

Winding Drum.—Cylindrical or conical. Flat rope winds on itself in a vertical plane. As the rope leading to the head gear is always in the same plane, a drum for a flat rope may be placed near a shaft,—say 20 yards distant. Drums of large diameter are used with wire rope.

The diameter of a drum ought not to be less than that of the winding pulley. Drums of too small diameters shorten the life of a rope. Drums are covered with wooden battens on which the rope coils.

With 10 ft. as a minimum diameter for a drum for a rope 1 in. circumference André tells as to add 6 in. to the diameter for every ¼ in. increase in the circumference of the rope.

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To determine the dimensions of shafts and axles, and distances apart at which supports may be placed and the diameter of a winding barrel for flat ropes (see Molesworth's Pocket Book).

Brakes.—A common form of brake is a series of blocks of wood which are brought into contact with the drum by means of a system of levers. With a constant pressure the friction of the brakes on the drum is theoretically independent of the extent of contact surface. A single block of wood, in which holes are bored and filled with sand, brought into contact by means of a system of levers is very effective. Steam brakes and brakes which act automatically are sometimes employed.

Rope.—Round or flat. Theoretically they ought to be tapered but tapered rope is expensive to make and difficult to coil.

Burat gives the following as an example of the economy of a taper rope. A rope 375 m. long was required to support 4,000 kilos. The whole length was divided into 3 section each 125 m. long with weights as follows,—688, 750 and 812 kilos. The total weight was therefore 2,250 kilos. If the rope had been constant in section in would have weighed 8 kilos per meter, with a total weight, of 3,000 kilos.

The gain was therefore 750 kilos or on two ropes 15,00 kilos and if the cost was 1.5 francs per kilo, the *money* gain was 2,250 francs, and as a rope may last 18 months the gain per year was 1,500 francs.

To Calculate Dimensions of Taper Ropes (*West*).—

If G is the girth at the top, g the girth at the bottom (calculated by ordinary rules), F the length in fathoms, then,

$$\text{For } \textit{steel} \text{ taper rope } \log G = \frac{F}{3{,}686} \times \log g.$$

$$\text{For } \textit{iron} \text{ taper rope } \log G = \frac{F}{2{,}060} \times \log g.$$

A steel rope may last 2 or 3 years. In a record respecting the wear of ropes, the size, length, cost of material, number of windings per day, maximum load, total load, durability &c. are noted.

The rope is fixed to the drum by taking two or three turns round the drum and then passing it through a suitable opening to the inside where it is coiled on the shaft.

The attachment of the rope to the *bridle chains* is by a link and *capping*. The methods of attaching a rope to a link by means of an iron cap is an operation deserving close attention.

Ropes often yield near the cage. To avoid the effects of jerking they are attached to the *bridle chains* by a strong spring. Iron ropes rapidly corrode in a ventilating shaft.

In ordinary wire ropes the wear is in the crown of the strand when there is the most bending. In Lang's patent rope owing to the principle of making a longer surface of wire exposed to friction the wear is consequently more distributed.

Flat ropes can be wound on a smaller drum than round ropes Their strength is approximately equal to the sum of the strength of the round ropes of which they are composed less 10 per cent.

Hempen ropes are heavy and their weight is increased by moisture.

The breaking strength of ropes for each pound per fathom is approximately,—

Hemp ropes	1 ton.
Iron ropes	2 „
Steel ropes	3 „

If the breaking strain in tons $= W$ and the circumference in inches $= C$, then for

Hemp ropes	$W = .25\ C^2$
Iron wire ropes	$W = 1.50\ C^2$
Crucible steel ropes	$W = 3\ C^2$
Plough steel ropes	$W = 4\ C^2$

The *proof strain* is $\frac{1}{3}$ to $\frac{1}{2}$ the breaking strain or twice the *working load* which is $\frac{1}{6}$ to $\frac{1}{10}$ of the breaking strain.

If W is the weight of a rope in lbs. per fathom then for

Hemp ropes	$W = \frac{C^2}{4}$
Iron or steel	$W = \frac{C^2}{1.2}$

(For tables of strength &c. see Molesworth.)

The gain in lightness by using steel ropes is in deep shafts very considerable.

Broken ropes are joined by a *shackle* or by a splice.

As an illustration of the limit to which ropes may be used take the case of a 2⅛ inch steel rope weighing 2 lbs. per foot and with a breaking strength of 84,000 lbs. This ought to sustain 42,000 feet of its the own length but for safety only 6,000 feet. If it carries a load of 3 tons the safe maximum length would be 3,000 feet.

Chains.—If the breaking strain in tons = W and D the diameter in sixteenths of an inch.

$$W = \frac{D^2}{9}.$$ Factor of safety 10.

Guides.—Usually of wood 4 inches by 4 inches, costing 12/- to 15/- per fathom, sometimes bridge rails, angle iron, round rods, or wire rope. The latter is usually stretched by a weight of 2 or 3 tons hanging in the sump. The clearance for cages should not be less than 9 inches. The *slides*, *cheeks* or *guide slippers* on the cage which clasp the guides on three sides are bell mouthed above and below. With guides of wood the slides are malleable cast iron, but with rope guides they are of brass. When iron guides are used, allowance must be made for expansion and contraction.

Wooden guides of oak or pitch pine may be in lengths of about 12 feet united by scarf joints and wedged to transverse timbers about 6 feet apart. If bolted to the transverse pieces, the bolt heads must be countersunk. With guides a cage may travel at rates up to 5 feet per second.

Keeps.—*Landing dogs*, *keeps*, *fans*, *shuts* or *chairs*, consist of a series of levers at the pits mouth, which are thrown back by the ascending cage. After the cage has passed, they return by the action of a spring or simply fall into their original position and the cage is lowered upon them, where it rests while the tubs are being exchanged.

The keeps below support the top deck of a cage.

The plummer blocks carrying the keeps ought to be bedded on some elastic material. This would reduce the severity of the shocks a cage receives when it is lowered on the keeps.

Cages.—These are usually of wrought iron. A single decked cage for one tub weighs 5 to 6 cwts., for two tubs 10 cwts. At Monkwear-

mouth a double decked cage for four tubs weighs 24 cwts. At the Grand Hornu Colliery a four decked cage for 8 tubs weighs 2,692 lbs. The lightest and best cages are made of mild steel; their weight being ⅓ to ½ of the gross load (tubs and coal). An iron cage weighs ⅔ of its load.

At the Pleasley Colliery double decked steel cages carrying 2 tubs on each deck weigh 53 cwts. Cages carrying one tub on each deck weigh 29 cwts.

Cages with several decks may be loaded or unloaded at a series of platforms corresponding in position to the height of the decks.

Cages are suspended by 3 or 4 or even 6 *bridle chains*. For safety, cage chains in the north of England are annealed once a month. To make these chains of exactly equal length so that they may equally share the load they may be altered in length by twisting or by a screw at the point where they connect with the cage. Catches which fall into position are employed to keep the tubs in the cage, or a false bottom in the cage which sinks an inch or two, may be used.

To Find Meetings with Flat Ropes (*Merivale*).—

Let n = half the number of revolutions.
d = distance of meetings from bottom of pit in inches.
r = radius of drum at lift in inches $+\frac{1}{2}t$.
t = thickness of rope in inches.

Then $$d = 3.1416n\,(2r + \overline{n-1}t).$$

Let D = depth of pit in inches.
n = number of revolutions.
r = radius of drum at lift in inches $+\frac{1}{2}t$.
t = thickness of rope in inches.

$$D = 3.1416n\,(2r + \overline{n-1}t).$$

$$n = \sqrt{\left(\frac{r}{t} - \frac{1}{2}\right)^2 + \frac{D}{3.1416t}} - \left(\frac{r}{t} - \frac{1}{2}\right).$$

$$r = \frac{D - \{n(n-1)3.1416t\}}{2 \times 3.1416n}$$

Balancing Loads.—

The forces acting on a drum shaft in opposite directions are often very unequal. On one side we have a cage with full tubs together

with the weight of the rope in the shaft (W_1) acting with a leverage equal to the radius of the drum on which it is coiled (R_1). On the other side tending to turn the shaft in an opposite direction we have a cage with empty tubs W_2 acting on the shaft with a leverage equal to the radius of its drum R_2. To balance, the loads $W_1 \times R_1$ must equal $W_2 \times R_2$. Now as the full tubs ascend the rope becomes short and the quantity $W_1 \times R_1$ become less, while the rope carrying the empty tubs becoming longer the quantity $W_2 \times R_2$ becomes greater.

After $W_2 \times R_2$ has sufficiently exceeded $W_1 \times R_1$, it will tend to pull the engine round and if steam were cut off the winding would be continued. Even if $W_2 \times R_2$ remains less than $W_1 \times R_1$ steam may be cut off near the end of a journey and the winding will be completed by the inertia of the moving loads.

The variation in loads here indicated may be more or less equalized by the following methods.

1. Use flat ropes coiling on themselves. As the cage ascends R_1 gradually becomes larger and may approximately keep the quantity $R_1 \times W_1$ constant. While for the descending cage R_2 becomes less and keeps the quantity $R_2 \times W_2$ approximately constant.

2. By using round rope coiling in the grove of a conical drum. For the ascending load R_1 becomes greater while for the descending load R_2 becomes less.

3. By a chain so connected with the engine shaft that by means of a series of weights attached at its other end which hang in a pit, it tends at the commencement of the journey to lift the full load in the shaft. As weight after weight reach the bottom of the pit, the assistance given to the engine becomes less and less. When all the weights have reached the bottom and the chain has run out, it commences to retard the engine which retardation is increased as weight by weight is lifted. This form is common.

4. Instead of a weighted chain, a rope or chain with a loaded tub running down and up a properly formed incline may be used as at Killingworth Colliery.

5. Employ a *tail rope* connecting the bottom of the ascending and descending cages hanging in the sump. In this case the winding rope carries the weight of the balance rope. A better form is to have a balance rope leading from the winding drums and guided by special pulleys down special compartments in the shaft. With this latter

arrangement the shackle of the winding rope only carries the cage and its contents.—(*Lindenberg and Meinickes System*).

6. A pendulum counterbalance as at the Dudley Colliery. This may be used for shallow shafts.

7. Engines with automatic variable expansion gear cause the power to vary with the load to be overcome.

Examples of Winding Loads (*Burat*).—

1. A drum is 6 m. diam. Length of rope 400 m. Weight of rope 7 k. per meter. Useful load 1,500 kilos. Dead load 2,000 kilos.

Then the moment at starting is $(1{,}500+2{,}000+2{,}800)\,3-2{,}000\times 3 = 12{,}900$ kilos. And at the end of the journey $(1{,}500+2{,}000)\,3-(2{,}000+2{,}800)\,3=-3{,}900$ kilos.

In this case towards the end of the lift the moment being negative the winding is effected without the aid of the engine.

2. At the Robiac Shaft at Bességes the depth is 400 m. Cylindrical drum 3.60 m. diam. Two pulleys of the same diameter on movable guides. Load 1,850 k. Cage 1,800 k. or total load 2,750 k. Round and conical rope 402 m. with a working strength ¼ its breaking strength. In this case the moments are:

At starting.....	2,500 k.
Just lifting the load	5,835 „
At the middle	3,358 „
At the end.....	880 „

The drum makes 35 turns and at the end the moment is positive.

The moments may vary with the age of a rope. If a new rope has a diameter $^1/_{10}$ greater than an old rope, then the initial radii of the drum with old and new will vary and the balancing be altered.

With flat ropes the biggest initial diameter and the smallest diameter of rope are the most favourable especially for a deep pit.

Calculations relating to the Balancing of Loads.

1. Counter balancing on an inclined plane.

Let P = weight on inclined plane.
a = angle of inclination of the plane.

R = radius of drum carrying P.
r = radius of winding drum.
w = weight of winding rope in lbs. per foot.
W_0 = weight of empty cage.
W = load.
H = depth of pit.
x = height of cage from bottom at the time t.
m_1 = moment exerted by the engine.

Then $\left\{ \begin{matrix} W_0+W+(H-x)\,w- \\ (W_0+xw) \end{matrix} \right\} r$ = moment resisting the engine.

This must equal the moment exerted by the engine plus the force exerted on the inclined plane, or.

$$R\,P \sin a + m_1 = r\,(W+(H-x)\,w-w\,x).$$
$$= r\,(W+(H-2\,x)\,w).$$

From which equation given R and P, the inclination of the plane can be found corresponding to any position of the cages.

1. Counter balancing by conical drums, one of which has a radius r_1 when the radius of the other is r_2. With values as above, for the same values of x the following two equations must be true.

(1). $\{W_0+W+(H-x)\,w\}\,x_1-(W_0+x\,w)\,r_2=m_1.$

(2). $\{W_0+W+x\,w\}\,r_2-\{W_0+(H-x)\,w\}\,r_1=m_1.$

$$\therefore \quad W r_1+W r_2=2m_1.$$

$$r_1+r_2=\frac{2m_1}{W}$$

Substituting in (1).—

$$\{W_0+W+(H-r)\,w\}\,r_1-(W_0+x\,w)\left(\frac{2m_1}{W}-r_1\right)=m_1.$$

$$\therefore \quad (2\,W_0+W+H)\,r_1=m_1 r\left(\frac{2W_0}{W}+2x\frac{w}{W}+1\right).$$

$$(2\,W_0+W+H\,w)\,r_1=\frac{m_1}{W}.\,(2W_0+W+2\,w\,x).$$

$$\therefore \quad r_1 = \frac{m^1}{W} \cdot \frac{2\,W_0 + W + 2\,w\,x}{2\,W_0 + W + H\,w}$$

$$r_2 = \frac{m_1}{W} \cdot \frac{2\,W_0 + W + 2\,H\,w - 2\,W\,x}{2\,W_0 + W + 2\,H\,w}$$

Pneumatic Hoisting.—This has been put into practice at the Epinac Colliery by M. Blanchet. By exhaustion, a piston in a tube is caused to rise and to this is attached the cage and tubs. The installation was expensive but it was chiefly abandoned because the workings were unremunerative. Up to 820 feet one ton of coal was raised by burning 16.5 lbs. coal. With cables for the same depth 55 lbs. were required.

With two tubes the efficiency increaces with depth and it may possibly be used for very deep mining. (Trans.: Am.: Mining Engineers. Vol. XIX. p. 115).

Man Engine and Endless Chain Hoisting.—The Méhu man engine was tried at Anzin and Ronchamp and the endless chain at Saint Jacques at Montlucon. They were expensive and dangerous and therefore abandoned.

Water Wheels.—Undershot, overshot, breast wheels, turbines and other pressure wheels.

Theoretical Horse Power.—

Q = quantity of water in cubic feet per minute.
h = head of water from tail race in feet.
P = theoretical horse power.

$$P = .001892\,Q\,h \qquad Q = \frac{528.5\,P}{h}$$

Memoranda.—

A cubic foot of water = 62.425 lbs.

p = pressure in lbs. per sq. inch.
h = head of water in feet.
V = theoretical velocity in feet per second.
g = force of gravity (32.2 feet).

Then $p = h \times .4335$. $h = p \times 2.307$.

Pressure per sq. foot $= h\ 62.4$.

$$V = \sqrt{2gh} = 8.025\sqrt{h}.$$

$$h = \frac{V^2}{2g} = .01555\ V^2$$

Effective horse power for different motors:—

Theoretical power being.....	=	1.00
Undershot wheel	„	.35
Poncelet's undershot wheel	„	.63
Breast wheel.....	„	.55
High breast	„	.60
Overshot wheel	„	.68
Turbine.....	„	.70
Hydraulic ram raising water	„	.60
Water pressure engine	„	.80
Knight and Pelton wheels	„	.82

In undershot wheels the power depends upon the square of the velocity of the water. To increase the velocity a stream may be narrowed in as it approaches the wheel. Power is gained by increasing the width of a float. Its depth must not be increased unless it is so arranged that it leaves the water in a direction perpendidular to its surface. The velocity of the periphery of the undershot water wheel should equal the theoretical velocity due to the head of water $\times$ 0.57.

h = head of water.

Q = quantity of water in cubic feet per minute.

P = effective horse power.

$$Q = \frac{1511\ P}{h} \qquad P = .00066\ Q\ h$$

In breast wheels the power depends upon the weight water.

In overshot wheels which are the most common, the power depends partly upon impact and partly upon weight. In order that water may readily enter a bucket it is necessary that there should be a suitable opening in the *shrouding* through which air may escape. (Ventilated buckets). The effect obtained among other things depends

upon the height at which the water enters the wheel and the shape of the buckets. According to Fairburn the diameter should be 1/8 more than the fall. The number of buckets should be twice the diameter reckoned in feet. A velocity of periphery for a fall of 5 feet is 7 feet per second. Ordinary wheels are 15 to 20 feet in diameter. At the Laxey Mine there is a wheel 72 feet 6 inches in diameter. These large wheels are difficult to construct and not so effective as two smaller wheels.

For Breast and Overshot Wheels.—

h = head of water in feet,
Q = quantity of water in cubic feet per minute.
P = effective horse power.

$Q = \frac{961\ P}{h}$ in low breast wheels.

$Q = \frac{881\ P}{h}$ in high „ „

$Q = \frac{777\ P}{h}$ in overshot „

$P = .00104\ Q\ h$ in low breast wheels.
$P = .00113\ Q\ h$ in high „ „
$P = .00128\ Q\ h$ in overshot „

Turbines.—Where there is a considerable head of water it is essential to use a Turbine. There are inward, outward, parallel and another class which may be called tangential flow wheels.

For formulæ relating to Jonval's low pressure turbine and Fourneyron's high pressure turbine see Molesworth.

Hurdy Gurdy, Knight or Pelton Wheels.—In these wheels which are simple in construction one or two jets under high pressure impinge against cups on the rim of a cast iron wheel. Their effeciency is said to exceed 82.6 per cent.

Men and Horses, Windlasses, Horse Whims.—Tackle or windlasses may be used to a depth of 20 fms. the kibble or bucket being made from boiler plate about 14″ high and 12″ diameter, and carrying 1 to 1¼ cwt. With horses a whim, derrick or whipsey derry may be used, the whim kibbles being 20″ to 24″ high and 14″ to 18″ wide and holding 4 to 6 cwt.

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For inclined shafts, skips running upon wheels are employed. The cost of a skip road or shaft railway is from £ 1.10 to £ 3.10 per fathom (*Collins*).

Three or four men at a windlass, do not usually cost more than a horse and its driver. A horse will do the work of about seven men. With horse whims, the main axle may be vertical or horizontal. Rather than employ three horses, it may be better to use a small engine. Materials may also be extracted from a mine by water wheels, pressure wheels, like turbines and Pelton wheels, hydraulic motors, by a water balance, &c.

Comparative Cost (*Collins*).

Two men with windlass, from a depth of 10 to 15 fms., raise 12 tons in 8 hours, at a cost of about ½ d. per ton-fathom.

One horse whim, from a depth of 40 fathoms, raises 15 to 20 tons in 8 hours, at a cost of about ⅙ d. per ton-fathom.

With engine power from 150 to 800 fathoms, 20 to 30 tons may be raised at a cost $^1/_{30}$ d. per ton fathom.

Winding by Water Balance.—Let a tank in the descending cage be loaded with until its weight is sufficient to raise the full tubs. The water is discharged at the bottom of the shaft and run off through an adit. At Montceau les Mines it has been applied to a depth of 225 feet.

Accidents in Shafts.—To prevent tubs running from the bank into the shaft, the mouth of the shaft is fenced with *gates* which slide vertically. When the cage comes up these are lifted. When the cage descends they fall down into their place.

In England between 1880 and 1890 there were 45 accidents resulting in 87 deaths, due to the breaking of ropes and chains in shafts.

To avoid such accidents, *safety catches* are attached to cages. So long as the cage is hanging by the rope, a strong spring or series of springs are compressed. If the rope should break, these springs immediately extend themselves, and their movement is used to force chisels or toothed eccentrics into contact with the guides which prevent the cage from falling.

At Anzin, *safety catches* have been used for a long time without accident, but it is thought that if the velocity of descent be high, the tension on the rope might be so far reduced that the catches might act when they were not required, resulting in greater or less damage to the guides.

They are largely used in the Australian Gold mines.

King and Humble's *safety grip* works upon guide ropes. If the winding rope is used to convey a current of electricity acting on electro magnets in the cage, these may be used as suggested by Mr. J. Yates to hold back griping cams, which if the rope should break must come into action.

It may here be mentioned that a current on the winding rope been tried in the Durham district to signal between the cage and the engine house.

Overwinding.—As the drums on which the rope is coiled are large, it might happen that by one revolution of the engine after the cage has reached the bank, the cage might be carried to the head gear. To prevent accidents of this kind, the engine man ought to have a clear view of the pit's month so that he can see a mark on the winding rope which indicates that the cage is near the surface. A *tell-tale* or *indicator* representing the movement of the cages in the shaft placed near the engine driver tells him at any instant the position of the cage. The movement of the model is derived from the winding engine. Sometimes the ascending cage is caused to ring a bell announcing its arrival near the surface. Sometimes the rope is attached to the cage by means of a *safety hook* (King and Humbles). In case of over-winding the hook is partially pulled through an opening in a set of timbers crossing the head gear. The lower part of the hook being large it is unable to pass the opening. It therefore comes forcibly in contact with it. By this contact the upper part of the hook is opened and the rope is freed. At the same time two projecting pieces are thrown out laterally and catch on the upper side of the cross piece and the cage is left suspended.

Another device to prevent over-winding is the use of a steam brake which comes into action after a certain number of revolutions of the engine.

PUMPING.

Martin found in Staffordshire that in 125 sq. miles area the quantity of water pumped was 50,000,000 gals or 220,000 tons in 24 hours. That is to say 10 times the quantity of coal which was raised in that district. The force required was 5,000 H. P.

In other districts, even when mines run beneath the sea, the water to be extracted may be very small. Water may be kept out of some mines by surface drains, or by tubbing off *feeders*. The temperature or the chemical character of underground water will sometimes indicate the source from which it comes. Limestone districts are usually watery.

To measure the water from a *feeder*, build a dam across the drift and let the water escape through a rectangular notch cut in a thin plate of metal, then

If G=gallons per minute.
d=depth in inches of the sill of the notch below the surface of the water.
l=length of notch in inches.

Then $G = 2.67\, l\, d\, \sqrt{d}$

A more certain measure of the water from a feeder is to determine the same from the time taken to fill a box of known capacity.

Dams.—Dams are used to prevent water passing from one portion of a mine to another. If of timber, they are built of wedge shaped blocks 3 to 8 feet long, put together as an arch of 18 to 30 feet radius, the back of the arch being on the side from which the pressure comes. Between the blocks there may be tarred flannel and wedges. They are built on a bed cut in the solid rock, in a narrow part of the level. During building, a large central man hole 18 inches in diameter is left for the ingress of workmen, a hole below for the escape of water and a hole above, 1 inch in diam for the escape of air. On completion, the lower hole is closed, then the man hole by drawing inwards a wedge shaped block, and finally, after water escapes above and the air has been expelled, the upper hole is closed.

Wooden dams will withstand 30-50 yards head of water. Brick dams are made with two arches about 2 feet apart, the space between being filled with clay.

Dams, Tubbing, &c.—

k=thickness in inches.
r=external radius in inches.
T=ultimate crushing strength in lbs. per square inch.
p=head of water in lbs. per square inch.

Then for cylindrical dam, walling or tubbing.

$$k = r\left\{1 - \sqrt{1 - \frac{20p}{T}}\right\}$$

Spherical Dam.—

$$k = r\left\{1 - \sqrt[3]{1 - \frac{15p}{T}}\right\}$$

A factor of safety of 10 is allowed for in the formula.

The values for T are as follows :—

Wrought iron	38,080
Cast iron	107,520
Beech	8,500
Oak	10,000
Pitch pine	6,500
Brick ordinary red	800
„ Stourbridge fire	1,717
Sandstone	2,185 to 7,884
Concrete	2,000

(*Molesworth*).

Raising Water by Simple means.—At the Wallesend and Hebburn collieries barrels of sheet iron 16 feet long, 3 feet 4 inches in diameter, each holding 800 galls, were used. Below they had a large valve which allowed them to be readily filled and readily emptied.

By this means as much 3,000 galls of water have been raised per minute.

In a mine 300-400 feet in depth a kibble of 2 cubic meters content, making 40 journeys per hour, will raise 80 c. m. per hour or 640 c. m. per day of 8 hours (Burat).

At mines worked by primitive methods small, hand pumps are used, each having a lift of 3 or 4 feet. In Japan these are 12 feet long, 4 inches diameter and have a stroke of 2 feet 8 inches. Sometimes Egyptian wheels about 8 feet diameter are employed. Rag and chain pumps are simple contrivances.

Water may be raised and carried over a barrier by a *siphon*, at the end of which there ought to be valves so arranged that when the water falls too low they are closed by floats, and the siphon remains full. Rotary pumps are good for lifting muddy water.

Suction pumps theoretically draw up water to about 32 feet (a height which varies with the height of the barometer) but on account of friction, slip, imperfect vacuum &c., they only draw to about 25 feet.

Force pumps theoretically work to any height, but when they exceed 100 feet the valves are subjected to great strains.

The lift of a so called suction pump may be 30–40 meters and sometimes even 70 meters (Burat).

At water works, compound reciprocating engines with double acting pumps are used. The duty reduced to 112 lbs. is from 95 to 100 million or even more foot lbs. With Cornish engines the duty is from 50 to 70 million foot lbs.

Cornish Pumps.—For the first lift commencing at the sump, a suction pump is used. Above this are a series of force pumps, all of which are worked by "*off sets*" from one set of rods.

In force pumps the plunger may either entirely fill the case, or the space in the case may be approximately equal to the solid content of the plunger. In this latter case, the flow of water may be made continuous. The plunger is made of iron or bronze.

Main, Rods, Pump Spears, Rocking Beams, &c.—Rods are usually of pitch pine. Theoretically they should be tapering. The lengths of pine are joined by scarf joints, wrought iron strapping plates and bolts. At Wheal Vor, which is 300 fathoms deep, we have the following :—

From surface to	780 ft.	2 timbers.	each 12″ square.
„ 780 „	864 „	1 timber.	16″ „
„ 864 „	984 „	„	14″ „
„ 984 „	1044 „	„	13 „
„ 1044 to bottom		„	12 „

As the Consolidated copper mines Gwennap, Cornwall, the main rod was one third of a mile long and weighed 95 tons. The other rods weighed 40 tons; 39 tons of this balanced the water in the pumps and the remaining 96 tons were balanced by counter weights &c. (*Collins*).

Iron rods do not work so smoothly as wooden rods.

Guides are employed at intervals as with the rods of a man engine.

The engine, which is single acting, raises the rods, and these falling by their weight, which is about 1/10 greater than the corresponding water column, raise the water. Any excess of weight is taken off by *balance bobs* placed at intervals in the shaft.

The main rods are worked by a rocking beam which may be of wood, wrought or cast iron.

In 1861 a cast iron beam at the New Hartley Colliery 34 feet long and weighing 42 tons broke, and carrying in the sides of the shaft as it fell, buried all who were below.

At Tresavean the rods and balance bobs were as follows.

Rods 16–17 tons.
4 balance bobs 16 tons each.
4 loaded balance bobs 18 tons each.

These with the water which is raised, represent 260 tons of material which is in motion at each stroke.

A *counter balance* framed like a triangle with the apex downwards carrying two balance weights has sometimes been employed. When the rods commence to fall, a large weight at the apex assists the rods, causing them to fall for a short distance very quickly and thus reducing the *slip*.

Hydraulic balances are sometimes employed to reduce the weight of the rods. They consist of a plunger which as it descends lifts a column of water. A plunger compressing air may be similarly employed.

In case of breakage, *indoor* and *outdoor catches* are placed at intervals down the shaft (see Man engine rods).

Valves.—Valves should be light and open to about an angle of 70°. If they open more, the slip, which is usually from 5 to 20 %, will be increased, and if less, the free entrance of the water is impeded.

Classification of Valves.—Michell in his "Mine Drainage," gives the following.

I.—Metallic Valves.

a. Ordinary clack valves.
b. Clack valves with contrivances to facilitate the exit of water.

2. Valves with vertical lift.

a. Single beat. *b.* Double beat.
c. Treble beat. *d.* Qnadruple beat.
e. Six beat.

II.—India Rubber Valves.

1. Flat valves.
2. Conical valves.
3. Lip valves.

Clack valves are almost entirely used in Cornish pumps.

The concussion in closing is great if the lift is high.

The slip is also great; sometimes 10 %. They are not suitable for steam pumps.

Those with contrivances to facilitate the exit of water are not suitable for high speeds. They do not close with sufficient rapidity.

Among single beat valves we have *cup valves*, *wing* or *mitre valves*, *spill* or *mushroom valves* and *ball valves*. All these valves have a high lift. Excepting the ball valve, they are suitable for light lifts. For high speeds they should have an India rubber spring and a large area compared with the working barrel. Double beat valves, like Harvey and Wests, have a large lifting area, a large water way and a low lift. The concussion and slip are therefore small. They are used for water works, but they are unsuitable for dirty mine water. An India rubber spring gives a quick close. Treble beat and other similar valves are still less suitable for dirty water.

India Rubber valves which are circular discs resting on a grating, and opening round the circumference, are suitable for pressures not exceeding 50 lbs.

Massey's valve, which has a revolving disc, has been used for pressures up to 150 lbs. per square inch.

Valves are made of wood, leather or India rubber.

In suction (lifting) pumps, the area of the upper valve is about one half the area of the cross section of the barrel. On the lowest suction valve a ring is placed, so that it can be raised for examination.

To avoid a shock on the valve a slight pause is made before commencing the downward stroke. The shock may be mitigated by dividing a valve into 3 or 4 parts. (See Michell's Mine Drainage).

Stroke and Lift.—The stroke is usually 2–10 feet and the lift may be 70 to 130 feet. A small diameter and long stroke is preferable to a large-diameter and short stroke, as the slip is less. In Cornwall the lifts rarely exceed 300 feet. They are usually 30 to 40 fathoms. At Creusot the lifts are 80–92 m., the lift depends greatly on the strength of the valves.

Callon describes a pump with a lift of 300 m. (328 yards).

Burat says, to raise 1,000 c. m. per day, the stroke should be 2 m., for 2,000 c. m. it should be 3 m., and for 3,000 m. 4 m. stroke. Long strokes give difficulty in the management of balance bobs &c. and necessitate a lower speed of working.

The following shews the advantage of a long stroke.

1. With a 4 m. stroke.

Rising	4	sec.	Total 16 sec. or 4 strokes per minute.
Descending	7	„	
Repose	5	„	

2. With a 2 m. stroke.

Rising	2	sec.	Total 11 sec. or 6 strokes per minute.
Descending	4	„	
Repose	5	„	

In the former we get 16 meters raised per minute while in the latter only 12 meters. In exceptional cases there may be 12 strokes. Five or six strokes are however more common.

In ordinary force pumps the water does not rise in a straight line. To obviate this a hollow cylindrical plunger with a double beat valve at one end has been designed. By this arrangement the water rises in a straight line through the plunger. The disadvantages are the difficulties of construction, and it can only be used for a small heights.

Pipes.—Pipes are usually of cast iron, but in case of lifting pumps sometimes of wrought. They may be galvanized.

If the water of the mine is acid, the inside of the pipes may be painted and then lined with wood. Under such circumstances the principal parts of the pump may be made of gun metal.

For cast iron pipes, if T = thickness in inches, D = diameter of pipe be inches, H head of water feet that will burst the pipe, then,—

$$H = \frac{72000\ T}{D}$$

If W = weight in lbs. per foot of pipe, D = outside diameter in inches and d = inside diameter in inches.

$$W = 2.45\ (D^2 - d^2).$$

Two flanges equal the weight of 1 foot of pipe.

An ordinary length for pipes is 9 feet. Internal diameters 9–10 inches and ⅞ inch to ½ inch thick. They are tested to 10 or 20 atmospheres. In ordinary joints, between the unplaned flanges, a ring of wrought iron covered with tarred flannel is placed. Flanges with planed faces do not require the ring. Sometimes a ring of lead wrapped with lamp cotton or a ring of gutta percha is employed. Specially formed flanges, where a groove has been cut for packing, withstand pressures of from 400 to 1000 lbs. per square inch. (See Michell's "Mine Drainage").

The velocity of water in pipes is usually 3 or 4 feet per second. Pipes should be straight and without contractions.

The thickness varies with the diameter.

Friction in Pipes.—

Diameter of Pipe in inch.	Head in feet per Mile required to balance friction.	
	For velocities of 2 feet per second.	
	(Boulton & Watt).	(Weisbach).
3	36.00	34.79
6	18.00	17.37
12	9.00	8.71
24	4.50	4.33

For velocities of 3 feet per second.

3	81.00	71.28
6	40.50	35.85
12	20.25	17.89
24	10.12	8.98

For velocities of 7 feet per second.

3	441	335.28
6	220.50	197.70
12	110.25	83.95
24	55.12	41.82

A general formula given by Merivale is

H = head of water in feet required to overcome resistance.
G = gallons per minute.
L = length of pipe in yards.
D = diameter of pipe in inches.

$$H = \frac{G^2 L}{(3D)^5}$$

Classification of Pumps.—Michell gives the following classification of steam pumps.

I. Horizontal Engines.

A. Rotary.

(Expansive condensing engines with connecting rod crank and fly wheel).
a. Simple steam pumps.
b. Compound steam pumps.

B. Non-Rotary.

a. Simple steam pumps. (Non expansive, non-condensing like the "Special").
b. Compound steam pumps. (Expansive, non-condensing).

II. Vertical Engines.

A. Rotary.

a. Simple steam pumps. (Expansive non-condensing).

B. Non-Rotary.

b. Simply steam pumps. (Non-expansive, non-condensing).

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It will be observed that all pumps with their engines in the above classification form one machine, whereas the Cornish engine and its pumps are separate machines. The Bull engine is a Cornish engine working directly over a shaft and therefore direct acting. In this case the pumps are also adjuncts to the engine. The objection to Bull engines is that they block up the shaft.

Fuel.—

A cornish engine requires about 3.5 lbs. of coal per H.P.
A common „ „ „ 5 „ „ „ „ „
A high pressure engine requires about 10 lbs. of coal per H.P.
A small high pressure engine requires about 15 to 20 lbs. of coal per H.P.

Duty of Pumps (*Michell*).

At the Tynewydd Collierya "Special" (steam cylinder 12," water cylinder 10" stroke 18") consumed about 14 cwts. coal to raise 300 tons of water 66 feet. This equals a duty of 3,168,000 lbs. raised 1 foot by 1 cwt. of coal and allowing for loss of heat say 4,000,000 lbs. which is 1/20 of the duty of a Cornish engine. A Cornish engine will give a duty of 95,750,000 lbs. raised 1 foot, by burning 1 bushel of coal (94 lbs.) or 114,000,000 lbs. for 1 cwt. of coal, or 33 times that performed by the above "Special." Mr. John B. Simpson assumes that a "special" requires 10–12 lbs. of coal per effective H.P. per hour. Michell says that underground it may require 15 lbs. This is equivalent to a duty per cwt. of coal of 14,784,000 feet lbs. which does not agree with the above statement of 4,000,000 lbs. Further, the duty of a Cornish Engine appears to be too high.

A "Special" has lifted 120 galls per minute 1040 feet and has worked for 2 years under heads of 500 feet. (*Michell*).

The compound differential pumping engine (Davey's Patent) works with high expansion and great economy.

There is usually considerable loss in conveying steam from a boiler on the surface to a pump of the Special type placed under ground.

Hydraulic Engines.—Hydraulic Engines are used for drawing water from dip workings or places where an ordinary engine might be submerged. The principle on which they work is to cause a small quantity of water under the pressure due to a considerable head, to aise a large volume of water a small height. The difference between

a hydraulic engine and a steam engine lies chiefly in the construction and method of actuating the slide valve. The hydraulic engines of Davey, Carrett Marshall are well known.

The pressure available in working such pumps is evidently the difference between the pressure in the pressure pipe and that of the back pressure in the delivery pipes. This principle is inverted in the Hydraulic Ram.

At the Comstock Lode, Nevada, a pair of hydraulic engines on the 2400 feet level, raise water 800 feet to the Sutro Tunnel. They are worked by water columns from the surface which are actuated by plunger pumps driven by a compound engine.

Water ejectors may be used for heights up to 100 feet.

Pulsometer.—A pulsometer works to a height of about 60 feet, and dirty water does not interfere with its action.

Horse Power for Pumping.—(*Molesworth*).

G = number of gallons to be raised in 24 hours.
F = number of cubic feet to be raised in 24 hours.
h = height in feet to which the water is to be raised.
HP = actual horse power required.

$$HP = \frac{G \times h}{4752000} \text{ or } \frac{F \times h}{762088}$$

25 per cent is added to overcome friction and 50 or 60 per cent more for contingencies, making a total of 70 or 80 per cent additional power.

Diameter of a Single Acting Pump.—(*Molesworth*).

L = length of stroke in feet.
G = number of gallons to be delivered per minute.
F = number of cubic feet to be delivered per minute.
N = number of strokes per minute.
D = diameter of pump in inches.

$$F = .00545\ D^2\ L\ N.$$
$$G = .034\ D^2\ L\ N.$$

$$D = \sqrt{\frac{G}{.034\ L\ N}}$$

$$D = \sqrt{\frac{G}{.00545\ L\ N}}$$

These formulæ give the net diameter of the plunger which is usually increased 1/4th to allow for leakage &c.

Comparison of Pumping Gear.—1. Pumping from the surface by means of rods. Here there is the theoretical objection that a large weight, a heavy mass of machinery and water, is continually being set in motion, first in one direction and then in another. The cost of the rods, balance bobs &c. is great and they encumber the shaft. The engine which may be compound, works with a high rate of expansion and is economical. Where Bull engines are employed, rocking beams &c. are avoided, and the first cost is less than with an ordinary Cornish engine, but the shaft is partially blocked.

2. Pumping from below. In this case the engine is badly placed and liable to accident from floods, falls &c. Rods are avoided and the cost is relatively small. Great pressures have to be provided against. The steam is not employed so economically as it is with surface engines.

LIGHTING.

Lighting usually costs from 1½ d. to 2½ d. per shift per man.

A primitive method is to use torches. The ordinary method is to use candles or lamps. Occasionally gas and the electric light is used. Petroleum wells may be lighted by reflection from the surface.

Candles.—The candles usually employed run from 20 to 50 to the lb. In Cornwall there are 12 to 16 per pound. The smaller candles are used where fire damp occurs. They may be carried in a lantern, on a holder or in a lump of clay. Unless carefully used there is much waste. 1½ to 2 lbs. may be used in three shifts of 8 hours each.

Oil.—In the south of France olive oil is employed while, in the north, colza is used. Sometimes walnut oil and fish oils are employed. The quantity used depends upon the size of the wick. Usually 4 oz. valued at 1½ d. to 2 d. are burnt per shift 8 of hours. A spherically formed tin lamp used in Scotland costs 2½ d. and it may be burnt for 7 or 8 hours for 1 d.

Safety Lamps.—All ordinary safety lamps consist of an oil lamp surrounded by fine wire gauze. An explosive gas entering the lamp takes fire inside, but because the wire is a good conductor of heat the flame can not pass to the outside until the gauze has become sufficiently heated, or it is forced through by a draught. In 1811 Clanny invented a safety lamp, which in 1815 was followed by the lamps of Davy and Stevenson, all of which embody the above principle.

Davy's Lamp.—In this lamp there are 28 wires to the linear inch or 784 apertures per square inch. It must not be placed in a current of air of 7 or 8 feet per second. Therefore it must not be swung or carried too quickly. It requires careful cleaning. This is done by heating the gauze in a muffle and then well washing. Weight 1½ lbs. and cost 7/. It gives about 3/10 of the total light. It may be so arranged that on opening it is extinguished. The lamp may be fastened with a lead rivet, or so arranged that the lock can only be opened by means of a powerful magnet. Davy's are used in the more dangerous workings.

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Stevenson's Lamp.—(The Geordie). This has a large cylinder of glass with gauze above. The air enters below. It is heavier than the Davy, but gives more light. It goes out when hot.

Clanny's Lamp.—This also has a glass cylinder but not so large as in Stevenson's lamp. It is heavy and brittle.

Mueseler's Lamp.—This is used in Belgium. It has a glass cylinder, a central horizontally placed gauze diaphragm, and a central metal chimney up which the heated gases ascend. It therefore burns well and gives a good light. It is easily extinguished in an ascending current or if inclined. It is one of the best lamps and its use is extending.

There are many other lamps but none of them are safe unless used with caution. When on fire inside they ought to be extinguished or quietly withdrawn to a pure atmosphere. Lamps with glass cylinders are liable to be accidentally broken or cracked by the firing of gas inside or by cold water dropping upon them. When the Evan Evan's lamp takes fire inside, a string is burnt, and springs are released which close the ingress and egress apertures for air, and the light is extinguished. In other lamps these openings are closed by hand and the lamp is extinguished by the products of combustion.

Davy, Clanny, and Stephenson lamps are respectively unsafe in currents of 400, 600 and 800 feet per minute. In mines, currents of 400 feet per minute are frequent and in main airways, currents approaching 2000 feet per minute have been recorded.

The Royal Commission on Accidents in Mines (1881) especially mentioned four lamps worthy of special attention. There were however others nearly if not quite as good. The lamps mentioned were.

Evan Thomas—This is a bonneted Clanny. It is safe in strong currents.

Marsaut lamp, which is a Clanny with three gauze caps.

Grays' Lamp is of the Eloin type.

Mueseler, with a shut off.

A lamp that has stood severe tests, which gives a good light and is self extinguishing in an explosive atmosphere is the *Thorneburry Safety Lamp.*

Before Davy's discovery, a *steel mill*, invented by Spedding in 1740 which gave light by means of the sparks produced by a steel disc revolving in contact with a flint was used. Explosions were however traced to its use.

Attempts have been made to use *phosphorescent bodies* like *calcium sulphide* and it is said that in S. America fire flies have been employed by miners.

For ordinary lighting, petroleum is sometimes used. It is cheap and in a pure still air gives a good light. The necessity of using glass chimneys or even of clock work to cause a draught is objectionable. Petroleum lamps can therefore only be used in special places.

Air forced through gasoline by a clock work arrangement gives a good light.

In some mines at the bottom of a shaft and along main levels, gas is employed. At Duffryn in S. Wales, the gas from a "*blower*" was collected and used on the surface.

Electric Lamps.—Electric incandescent lamps have been used at certain collieries in England and again in certain gold mines in Victoria One H. P. is required for 8 or 10 lamps each 20 candle power.

Portable electric lamps have been invented for working places, as for example.

1. A Geissler tube illuminated by a Bunsen cell and aRuhmkorff coil.

2. An incandescent lamp illuminated by six or more small bichromate elements.

3. An incandescent lamp illuminated by small accumulators. Mr. Swan's lamp weighs in all about 9 lbs. and measures 7½ in. by 4¼ in. It gives a light of 2 or 3 candle power and burns 11 hours. A 5 H.P. dynamo will charge 300 of these lamps at one operation.

Other electric lamps are Bristöl's portable safety lamp. Pitkin's portable batteries and hand lamps. Cathcart, Peto and Radford's lamps. Woodhouse and Rawson's lamps. These lamps usually employ secondary batteries. They weigh from 2 to 9 lbs.; give a light of from 1 to 4 candle power aud burn from 4 to 10 hours.

Fleuss Lamp.—In the *Fleuss lamp* the combustion is supported by a special supply of oxygen which drives the flame of a spirit lamp against a ball of lime.

Aerophores are practically air tight bags containing about 12 cubic feet of air which can be carried on the miner's back. The air is used for respiration and to keep a lamp alight, as for example, after an explosion. In the *Denayrouze* form, the air is compressed to 300 or 350 lbs per square inch.

VENTILATION.

Necessity for Ventilation.—That there is proper supply of air to a Mine is a point which must be attended to by every Mining Engineer.

Should the ventilation be insufficient the *health* of the workmen will be impaired. In many districts especially where mining operations are conducted on principles which are more or less primitive, evidence of this may be seen in the sickly appearance of the Miners. In other districts the same result, although not directly exhibited in the individual, is seen in the statistics of the death rate and the cost of *out put*, for men can neither work so well, nor live so long, in a vitiated atmosphere as they can in one where the air is plentiful and pure. In this way we see that the proper ventilation of a mine, becomes a means of lessening expense. It is also the means by which *explosive gases* like those which emanate from so many of our coal mines, are swept away, and sudden destruction of life and property are consequently avoided. Also in mines where Gunpowder and other explosives are used for blasting purposes, a quantity of smoke is invariably produced, which if not rapidly swept away becomes a serious impediment to the men in returning to their work, thus causing delay and consequently another source of expense. For reasons such as these, but more especially perhaps to avoid the loss of life, many governments have drawn up rules to insure proper ventilation in mines. Thus in certain parts of the United States, it is provided that air shall be supplied at the rate of 55 cubic feet per second for every 50 men.

Where much fire damp is present 70 cubic feet may be required per minute per man or 53 cubic feet per ton of coal extracted in 24 hours.

Causes vitiating the Air of a Mine.—The goodness or badness of the air in a mine depends to a great extent upon its composition. An average analysis of air in the climate of England, is as follows.

Nitrogen,	77.950	per cent.
Oxygen,	20.610	„ „
Water,	1.040	„ „
Carbonic anhydride,	.041	„ „

Of these constituents which are mechanically mingled together, oxygen the chief supporter of life and combustion, is the most important. If by any means it is diminished by only one per cent, the remaining air is no longer fit to be breathed. If it is reduced by two per cent, it is only just capable of supporting the flame of a candle, and if four per cent is taken away a lamp will refuse to burn, whilst air with five per cent less oxygen than that contained in the ordinary atmosphere can not be breathed.

Not only is air vitiated by the withdrawal of oxygen, but it is also vitiated by the addition of various deleterious gases. The most important of these are as follows.

Carbonic anhydride. — CO_2 S.G. 1.529. — In the ordinary atmosphere this gas exists in traces. One of its chief sources is from respiration. Air which passes through the lungs of a man loses from 3 to 4 per cent of its oxygen, nearly all of which is exhaled in the form of carbonic anhydride. For an ordinary individual it is calculated that 15 cubic feet of air per minute would be sufficient, but for a person actively engaged it has been estimated that nearly four times this quantity would be required. Accepting this latter estimate of 60 cubic feet per minute as being a liberal allowance for a working miner, and that 3½ per cent of this quantity is converted into carbonic anhydride, we have each man producing 2.1 cubic feet of this gas per minute. A horse produces about 6 times this quantity. This gas is also formed by the action of almost any feeble acid upon carbonates. During processes of fermentation and by the oxidation of organic matter in solution, whenever any body containing carbon is burnt, carbonic anhydride is also obtained.

Angus Smith estimated that the burning of two candles produces about the same quantity of carbonic anhydride as one man would produce during an equal interval of time. André however gives 2.5 cubic feet as being produced per hour by the burning of a candle, which is about the same quantity as that which is produced by the gnition of 1 lb. of blasting powder.

When mixed with air it forms one of the principal constituents of *choke damp*, a mixture of the deleterious gases which result from the explosion of carburetted hydrogen or *fire damp*. *Choke damp* is also known by the names *black damp*, *after damp*, *stythe* &c.

In coal mines especially in those where lignite is worked it is often found in quantity, but in other mines it occurs but seldom and then only in traces.

In the concentrated form it is irrespirable, producing spasm of the glottis. If the proportion of the gas in the air reaches 8 to 10 per cent lights refuse to burn. This power of extinguishing a light affords a test for its presence when in quantity, but it must be borne in mind that it is quite possible for a light to burn in an atmosphere containing a sufficient quantity of this gas to cause death. Three to four per cent of it will act as a narcotic poison, and even smaller quantities are very injurious.

In wells where CO_2 has accumulated men have lost their lives. At Pontgebaud in S. France machines could not keep it out. In Auvergne 17 cubic feet of this gas have escaped per minute from a surface of 10 feet and in another 1100 cubie feet in 10 minutes.

In the air from mines Dr. Angus Smith found on an average as much as .78 per cent, and in several extreme cases as much as 2 per cent of this gas. If it is necessary to enter an atmosphere containing this gas, as for instance after an explosion, Graham has suggested that the mouth and nostrils should be covered with a cloth containing a layer about 1 inch thick of moist lime and crystallized sodic sulphate, such a layer being porous enough to allow of respiration, whilst the lime absorbs the carbonic anhydride.

The power of this gas to extinguish flame has been applied by Sir Goldsworthy Gurney to extinguish conflagrations, such as those which occasionally break out in coal mines. In such cases all entrances to the mine excepting two were closed. One of these was used for the entrance of the gas, and the other for its exit. The gas was produced either by forcing a current of air over a furnace in which coke and charcoal were burning or else by acting on carbonate of lime with sulphuric acid. This was forced in to the mine with jets of steam, the water thus formed assisting to cool down the workings.
(see p. 100)

On account of its high specific gravity CO_2 will tend to accumulate in the lowest portions of a series of workings. But it is also often found to be more or less diffused through the air particularly when assisted by currents of air and varying temperatures. It is easily absorbed by water. The absorption is increased if the water contains slaked lime. Advantage can not be taken of this property for the removal of this gas, because the absorptive action has been found to be too local.

Carbonic Oxide. CO. S.G. .967.—This gas is often produced in quantity when combustion takes place, as for instance during the process of "*fire setting*." Like carbonic anhydride it is transparent and colourless, but unlike it, it is only absorbed in small quantities by water. As might be expected from its specific gravity it readily mingles with air. It is very poisonous and according to *Dumas* and *Leblanc*, if air contains only one per cent of this gas, it is sufficient when breathed to cause a miserable death. Although carbonic oxide is a non-supporter of combustion, a lamp may burn in an atmosphere containing a sufficiently large quantity to cause death.

In small quantities it produces a trembling dizziness which is followed by insensibility. Fortunately for the miner this gas is of rare occurrence. It was met with at the Hetton Colliery.

Methane, Marsh Gas or Carburetted Hydrogen. Gr. *dampf* or *schlagende wetter*. Fr. *grisou*. CH_4 S.G. .557.—This gas is sometimes formed by the spontaneous decomposition of vegetable matter beneath stagnant water. From this it has been called *marsh gas*. By processes of natural distillation it is produced and given off from many coal seams where you may often hear it escaping with a low hissing sound. Sometimes it is given off with great violence, having evidently been under pressure. Mr. Lindsay Wood observed at Boldon a pressure of 461 lbs per square inch. The fissures or orifices from which it thus escapes are called *blowers*. The length of time that a *blower* may continue to give off gas is very indefinite; it may exhaust itself in a few hours or it may even continue for years. In mines where large quantities of this gas are given off, it is necessary that bore holes be carried along in front of the miners lest a reservoir of it should be suddenly tapped. By this means it may often be slowly drained away, and the danger of a sudden explosion avoided. It is especially met with in coal which is soft and

friable. It occurs but seldom in brown coal. The largest quantity of gas is given off from newly made openings or from freshly broken coal. But although the quantity which is given off continually tends to decrease, it will slightly vary with the atmospheric pressure, the variation of a ventilating current, the increasing of the temperature of the mine. Owing to its lightness it tends to rise towards the highest workings, and almost invariably large quantities of it will accumulate in the *goaf*. It is estimated that the air space in *goaves* is one sixth of the volume of the coal removed. From these places it escapes in quantity by any variation in pressure.

As to whether there are large accumulations of gas in a bed from which it is known to be given off, depends largely upon the geological nature and position of the seam and the surrounding rocks. Sometimes a fault may act as a reservoir in which to store the gas, but on the other hand it may act as a canal to drain the gas away. This draining away of the gas may also occur when the surrounding rocks are wanting in compactness, or when the seam reaches up to the surface of the ground &c.

This gas is not only found in the coal, but also in the shales which surround the coal. In districts where substances like Naphtha and Asphalt are found it is likely to occur. In one or two instances it has been found in salt and metal mines, but such cases are rare, carburetted hydrogen being a gas which is essentially peculiar to coal mines.

In metal mines it has been met with in the Hartz, in Flintshire, Montgomeryshire: Okel Tor in Cornwall, at Lead Mines in Derbyshire, at Silver Islet. It has also been observed in iron mines.

It is a colourless inodorous gas, a non-supporter of combustion, but itself inflammable, burning with a blue flame.

When mixed with air, with which it is easily diffused, it constitutes "*fire damp*" an explosive mixture often met with in many coal mines.

If 3 volumes of carburetted hydrogen are mixed with 30 volumes of air, its presence is indicated by the flame of a lamp becoming long and thin. From appearances of this kind, as for example by the formation of a blue halo round the flame of a candle, the separation of the flame from the wick to form a '*top*' or '*cap*,' the miner receives warning of its presence. At one time to test whether gas of this nature was present in a mine, special men were employed, who before

the entrance of the workmen, advanced cautiously towards the dangerous places holding out before them a small thin taper. If any change in the flame was observed to be taking place they cautiously withdrew. Now the miner is warned by the enlargement of the flame in his safety lamp, which he then carefully extinguishes, either by drawing down the wick, or else by plunging it into water.

At the Oaks Colliery, Barnsley, gas suddenly came out along a length of 50 yards. All the Stephenson's lamps were put out. Davys' lamps became red hot but were extinguished by drawing down the wicks. The return air did not go over the furnace and in consequence of these precautions there was no explosion.

When the proportion between gas and air is as 1 : 12 a weak explosion will take place, but with the proportion of about 1 : 8 a violent explosion will occur. With the ratio 1 : 5 the explosion is more moderate, and with 1 : 3 all explosive power is lost.

The result of a powerful explosion is represented as follows :

20 vols of air or 2 vols of gas or "*after damp*"

$$8\,N_2 + O_2 + CH_4 = 8\,N_2 + CO_2 + 2\,H_2O.$$

Not only has the production of *after damp* or *stythe* proved fatal to those in the immediate vicinity of an explosion who may have been fortunate enough to have escaped the effects of mechanical violence, but also to others who have ventured to their assistance, or who may have been engaged at work in some portion of the mine to which this deleterious mixture has been subsequently carried.

By looking over a series of analyses of the gases which escape in coal mines, it seems that we may obtain mixtures of very variable compositions. We have generally from 77 to 98 per cent of carburetted hydrogen, a small percentage of oxygen, from .1 to 7 per cent of carbonic anhydride, from .5 to 17 per cent of nitrogen and a small quantity of aqueous vapour.

Sulphuretted Hydrogen. H_2S. S.G. 1.19.—This is a transparent colourless gas, which is easily recognized by its disgusting odour. When concentrated, it is highly poisonous. With 1500 times its bulk of air it is fatal to birds, and with 250 times its bulk, to larger animals, like horses. It takes fire at a low red heat. It is formed

spontaneously by the action of decaying animal or vegetable matter on a soluble sulphate. It is also formed by the explosion of gunpowder. Although explosions are on record as having been caused by this gas, it is of rare occurrence, and it is only in old workings, and rarely even there, that it is found in any quantity. It occurred in Collieries at Whitehaven.

Sulphurous Anhydride. SO_2. S.G. 2.24.—This gas which is often emitted from the craters of volcanos, is produced when sulphur is burnt. It is irrespirable and non-inflammable. It is produced whenever gunpowder is exploded.

Ammonia, H_3N. S.G. .59.—This gas is formed by the spontaneous decomposition of animal matter, such as the excrement of animals.

Arsenic and Mercury.—In mines where these metals are worked, their vapours are sometimes produced. Vapours of arsenic are easily removed by ventilation, but the removal of those of mercury is more difficult.

Dust, Aqueous Vapour &c.—In addition to the above mentioned gases, the air of a mine may be rendered impure by the addition of various mechanical impurities; thus for instance we often find that in coal mines, and especially in those which are very dry, dust is very plentiful. This is naturally a check upon the free respiration of the workmen, and should an explosion take place in an atmosphere thus charged, the effects are considerably intensified. Galloway has shown that an explosion may be caused by coal dust and air. The danger resulting from coal dust depends upon its fineness and on its inflammability.

If the atmosphere of a mine is charged with aqueous vapour, and more especially if this be accompanied with a high temperature, ill effects will be produced not only upon the workmen, but also upon the materials of the mine.

Determining the presence of Various Gases.—The presence of various gases of a deleterious nature in the workings of a mine, is to a careful observer, evidenced by many appearances. The health of the workmen, the way in which the lights burn, how the timber stands, and many other signs, often indicate the presence of a

greater or less quantity of obnoxious gas. When the quantity of any gas is large, its effects are more decided. Sometimes lights are suddenly extinguished, at other times explosions take place.

When air is not fresh and pure, either from the want of oxygen, or by the addition of deleterious gas, the flame of a lamp or candle, instead of being bright and clear, becomes smoky and sooty. Should *fire damp* be present, it burns blue; if carbonic anhydride is present, it may be extinguished.

One of the older plans to determine the presence of explosive gases in a mine, was as has already been mentioned when speaking of Carburetted Hydrogen, to employ special men to inspect the suspected districts by means of lighted tapers.

Another but not altogether practical plan which has been suggested is to have an electric circuit, broken at many points, running through the mine. By sending a current, it was possible at such places to cause sparks to pass, and thus ignite any explosive gases which might be present there. Although danger to the workmen might be thus avoided, the danger to the mine was too great to warrant its application, excepting under peculiar conditions.

The Turquau Detector.—To determine the presence of *fire damp*, Turquau a Frenchman, constructed an alarum, the stop of which was held by a string dipped in Saltpetre passing through the gauze of a safety lamp. Before an explosion took place, the gas ignited in the inside of the lamp, the string was burnt and the stop set free. The noise of the alarum gave notice to the miners who had at once to quickly leave the mine.

Osmoscopes.—A number of instruments have been constructed to take advantage of the diffusion of gases.

One form arranged by Ansell, is that of a bent tube, terminated at one end by a cup the top of which is covered with a lid or diaphragm of burnt clay or marble. At the other end there is a small globe, running through which there is a metal screw. This tube is filled with mercury which is in connection with one end of an electric circuit, the other end of which terminates in the above mentioned screw. If such an instrument as this be placed in an atmosphere of explosive gas, the gas will by diffusion pass more quickly through the marble or clay diaphragm than the air which is inside, can escape,

The consequence of this is that the mercury will be forced up the branch of the tube until it reaches the screw. The circuit being completed an electric bell will be set in action and the miners warned.

If the point of the screw be placed very near the surface of the mercury the warning will be given very soon. In this way, and also by altering the nature and thickness of the diaphragm the action of the instrument may be varied.

Another form consists of a bent tube passing through a cork covered with sealing wax into the interior of a porous cell. By diffusion, explosive gas enters the cell more quickly than the air escapes, and water is forced up one arm of the tube. On the other hand, should gas like carbonic anhydride be present, the air will come out quicker than this dense gas can enter and the coloured liquid will rise in the other arm.

Instruments of this type when suddenly immersed in an atmosphere which has a density different to the air they contain, work satisfactorily, but when the outside gas accumulates slowly as in a mine, *osmosis* is sufficiently rapid to keep the pressure inside the instrument practically equal to that outside and therefore as *fire damp* detectors these contrivances are useless.

A Whistle Detector.—Prof. George Forbes has devised a whistle the note of which varies with the density of the gas in which it is sounded.

A Balloon.—A small balloon floating in mine gas will exert more or less force upon a spring holding it captive according to the greater or less density of the atmosphere in which it floats.

In consequence of diffusion the value of such an arrangement would be quickly destroyed.

The Barometer and Thermometer.—The evolution of gas at a mine, especially from *goaves*, ought certainly to vary with atmospheric pressure, the greatest quantity escaping when the barometer in low. This is so far recognized that the Mines Regulation Act demands that a barometer and thermometer be placed near the entrance of a mine. That some engineers have failed to recognize the value of these instruments may be because they have only had experience at mines which are not sensitive—it being exceedingly probable that for the same reason that

it is only certain volcanoes and springs which show a close relationship with fluctuations in atmospheric pressure, it may also only be at certain mines where changes in the weather are closely connected with the giving out of gas. For a mining engineer to take full advantage of barometric changes he should be able to foretell such changes. To accomplish this a mining district requires to be in connection with a Meteorological Bureau and receive the same information that is furnished to sea ports. The use of the thermometer and hygrometer in conjunction with the barometer are obvious.

Earth Tremors and Mine Gas.—Several writers have drawn attention to the relationship of earthquakes and colliery explosions, while committees have been formed to determine whether there is a connection between the escape of gas and earth tremors. In England the instruments employed only recorded earthquakes. In France *tromometers* have been employed to record *earth tremors* but it is the author's opinion that none of the experiments were of a nature likely to yield the best results. The instruments required are those which record minute rapidly recurring *earth tilting* and these have hitherto only be used in Japan. *Earth pulsations* may be regarded as similar to the swell on an ocean. They are closely connected with barometrical changes, they outrace the wind and to determine whether they are connected with the escape of *fire damp* would be a legitimate investigation.

Lamp Detectors.—Pieler's alcohol lamp is a practical fire damp detector, the flame being very sensitive. In Livings' apparatus the light from two incandescent platinum spirals, one being in the air and the other in the mine gas, are compared by a photometer.

Swan's fire damp indicator acts on the principle that if a mixture of air and fire damp is burnt the volume of the mixture is diminished in proportion to the percentage of fire damp present, pressure and temperature remaining constant. The apparatus consists of a combustion tube inside which there is a platinum spiral which can be heated by the current from a battery. The diminution in volume is determined by the rise of liquid in a gauge tube.

Maurice's fire damp indicator works on similar principles. In Murday's fire damp detector, two platinum wires are heated by a current. One is exposed to the air of the mine and if gas is present it is more highly heated than the other which is enclosed in an air tight cylinder.

The result is a differential expansion, and this actuates a pointer which may be used to ring a bell.

Avoidance of the accumulation of deleterious Gases.—From what has been said of these various gases and vapours it will be seen that their development and accumulation may to a great extent be avoided by careful attention to the ordinary management of a mine.

To keep the air of a mine pure, the mine itself should be kept pure; bad timber, putrescible and excremental matter should be removed, the water channels should be kept clean, smoking and all unnecessary burning of material should be avoided. All old workings where bad gases may accumulate should be carefully closed. To do this it is well, where possible, to build either a stone, or else a brick and mortar wall, which may then be covered with asphalt. It is also advisable to work regularly for if this is not done it will often be found that many of the workings during the absence of the workmen gradually become filled with gas.

In sinking shafts and other special works it is sometimes found that gas which only accumulates slowly may be destroyed by leaving lighted lamps which decompose the gas as it is formed.

Various chemical means for destroying carburetted hydrogen and other gases have been tried, but chiefly on account of their local action, the nature of the resulting compounds, and the expense, these do not appear to have attained any great success.

Dust may be kept down by sprinkling crude salt in the roadways or by watering them.

Avoidance of Explosions due to Gas and Dust:—The danger of explosions due to the ignition of gas or dust may be minimized by the use of safety lamps, the firing of charges by electricity or friction caps, and the use of explosives incapable of causing flame. Ordinary dynamite in holes stemmed with dust has given a flame 95 feet in length and all high explosives act similarly. The Austrian fire damp Commission recommend the use of *Soda-wetter-dynamite*, which is a dynamite containing 32 per cent of crystallized carbonate of soda. Roburite, Securite, Carbonite &c. have also low temperatures of explosion (see p. 61). The same Commission also suggest that for safety the return air should not contain more than 1½ per cent

of fire-damp and ½ per cent of carbonic acid, and that the air in the mine should be frequently tested with a Pieler lamp.

Expulsion of Bad Air, and supplying of Fresh Air.—In most mines and especially in collieries it is found impossible to so-far check the formation and accumulation of bad gases that a systematic ventilation is unnecessary. At the end of levels, in small pits the miner will often remove an accumulation of bad air by waving to and fro a branch of leafy twigs or his jacket. This however is a method of ventilation which can only be applied to openings which are not of large extent. More generally however, in order to obtain a current of air through the workings of a mine we have to depend upon the relative positions of various openings and the judicious opening or closing of levels, or else to rely upon the action of some machine or contrivance by which a current of air produced artificially may be either drawn or forced through the various passages which we wish to ventilate. This leads to the consideration of the two principal methods by which a mine may be ventilated, namely *naturally or artificially*. In some cases a mine is altogether ventilated by either one or other of these methods, and sometimes by the two combined.

Natural Ventilation.—The natural ventilation of a mine depends to a large extent upon the difference in temperature above ground and beneath.

One cause producing this is the natural temperature of the earth. At a sufficient depth to escape the effects of superficial changes of temperature, the rate of increase is about 1° for every 50 feet of descent. In volcanic districts this gradient may be steeper.

This temperature is increased by the burning of gunpowder, by lights and by breathing of men and animals. In some mines it is augmented by the burning of combustible gases, while in mines like the copper mine of Beshi in Shikoku, by chemical decomposition. At this latter place even at the entrance to the mine, there is a temperature of 95° F. In the lower levels of the Comstock Lode the water has reached 170° *F.*

In many cases these natural temperatures are sufficient to produce a continuous current of air, but in other cases, either from time to time, or else continuously they will require more or less artificial assistance.

We find examples of natural ventilation in the hilly lands of Devon, Cornwall and in the Midland Districts.

Nicholas Wood gives the following instance of natural ventilation.

In the Seaham Collieries, Hetton Seam, a shaft 360 fms. drifts 336′ long. Diam: of pit 14′. Temp. on bank 44°, at bottom of downcast 49.° At base of upcast 52½°. The result was 7002 cubic feet of air per minute.

At the Tythe Pit 672 feet deep. Surface Temp. 43°; at the bottom of downcast 45°; the return air 63°. This gave 36,564 cubic feet per minute.

When air is heated it expands, becomes lighter and consequently tends to ascend, whilst the cooler and denser air sinks down to take its place. If we imagine a tall chimney to represent the shaft of a mine, the entrance to which is represented by an *adit* level, we see that just as the fire in a furnace produces an upward draft of air in the chimney, so would any internal heat of a similarly arranged mine also produce a current of air through its *adit* and up the shaft.

It *A B* is a shaft and *B C* is an *adit*. So long as the air in the mine is of the same temperature as the air outside there will be a column of air *A B* in the shaft of the same density as an imaginary column *D C* vertically above *C* on the outside, and the two will therefore balance each other. However should the air in the inside of the mine become warmer than that outside, the column *A B* in the shaft will be lighter than the column *D C* on the outside and will therefore be outweighed by it, and a current of air will set into the mine at *C* and come out at *A*.

In this case the shaft of the mine acts like the chimney. It may however happen that the air in the mine will be cooler than the air outside, in which case *A B* being heavier than *D C*, the column of air outside, in which case *A B* being heavier than *D C* the current of air will enter at the top of the shaft *A* and find an exit at *C*.

Force of a Ventilative Current.—To determine the force which produces the ventilating current we must determine the difference in weight between the column of air in the shaft, and that over the mouth of the level. This difference which is the moving force, is usually measured by what is called the *Head*.

The *head* is the height of a column of cold air whose weight is equal to the above force.

If T be the mean temperature of the shaft, and t that of the air outside, and if T is less than t, then the *head* in feet equals,

$$H\frac{t-T}{t+273} \text{ or } Ha\frac{t-T}{1+at}$$

whereas if T be greater than t then the moving, force or *head* equals,

$$H\frac{T-t}{T+273} \text{ or } Ha\frac{T-t}{1+aT}.*$$

In both these cases H equals the depth of the shaft or AB or CD of the preceding supposition, and $a=.00377$ or $\frac{1}{273}$ this being the increment produced by an elevation in temperature of one degree centigrade in the volume of gas at zero under a constant pressure.

In these expressions the *head* is expressed as a column of air so many feet in height. The same column might be expressed as a column of mercury or of water so many inches in height.

Instead of considering the case of a mine which is ventilated by means of a shaft and a level, we consider the case of a mine which is ventilated by means of two shafts, one of which called the *upcast*, being used for the exit of the air, and another called a *downcast* for its entrance, we arrive at similar result.

Farther, we observe that the greater we make the difference between the temperatures in the two shafts, that is, between T and t, the greater will be the moving force producing a ventilating current.

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* 1. Outside there is a column of air H of a temperature t.°

2. In the shaft there is a column of air H of a *higher* temperature T°. Let h be the height of the air in the shaft when at O°C.

Then $$h+a\,T\,h=H$$

or $$h=\frac{H}{1+a\,T}$$

The height in the shaft at t° will therefore be,

$$\frac{H}{1+a\,T}+\frac{H}{1+a\,T}\,a\,t=H\frac{1+a\,t}{1+a\,T}$$

and the difierence between this and the column H outside, which is of the same temperature is,

$$H-H\frac{1+a\,t}{1+a\,T} \text{ or } Ha\frac{(T-t)}{1+a\,T} \textit{ head} \text{ in feet.}$$

Because in a wet shaft, the air is generally cooler than in a dry one, such a shaft would usually be the one chosen for a *downcast*. It is for this reason that the shaft in which the pumps are placed is the one which is used for the *downcast*.

Measurement of Pressure.—From readings of the thermometer we see that it is possible to calculate the pressure of air producing a ventilative current. This is however usually observed directly by means of a Water Guage which is a bent glass tube, in the bend of which there is a small quantity of water. It is open at both ends.

By placing this in such a position, as, for instance, with one of its ends through a hole in a *stopping*, the current of air, passing in at the opening in one of its arms, forces the water up in the other. By graduations upon this latter arm, the pressure is read off as so many inches of water. We also see that barometric readings might indicate the pressure in millimeters of mercury.

If a barometer were placed near the top of an *upcast* and another at the entrance to an adit, this latter would stand higher than the former, and the difference between them would indicate in millimeters of mercury a portion of the moving force. If we had an *upcast* and a *downcast* situated near the same level, barometers placed near the entrances to these ought continually to give the same readings. They would however differ from the readings of barometers placed at the bottoms of two such shafts, and these latter would again differ from one another,—the one placed at the bottom of the *upcast* standing higher than the one at the bottom of the *downcast*.

As readings on a mercury column due to great differences in *Head* are only small, and also as they vary not only with the pressure but also with the velocity and temperature of the air, the barometer can not be so successfully employed as the Water Guage in the measurement of pressure. Anemometers of various descriptions also indicate pressure but they are usually employed to measure velocity.

Velocity of a Current.—If a body has fallen from a height h, it has a velocity which is equal to $\sqrt{2gh}$. It happens that the same formula gives us the velocity of a fluid: that is, if air is flowing from one place to another, the difference in pressure between the two places being measured as a head h, then the velocity v with which air flows to the place of lower pressure is approximately $\sqrt{2gh}$.

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The head h is the height of a column of air whose weight is the difference in pressure between the two places.

Having given the *head*, from this expression the velocity may be calculated. It may also be remarked that if the velocity has been observed, the *head*, which is $\frac{v^2}{2g}$ may be calculated.

Sometimes it will be convenient in the formula $v = \sqrt{2gh}$, to substitute for h its value $H a \left(\frac{T-t}{1+aT}\right)$ (see p. 176) and we obtain.

$$v = \left(2g\ H a \frac{T-t}{1+aT}\right)^{\frac{1}{2}}$$

To observe the velocity of a current anemometers are used. These are of two kinds, 1st those with a rotary motion, and 2nd those which act by the pressure of the passing air against a swinging vane.

Amongst the former, *Biram's* is perhaps one of the most common which is used in England. It consists of 12 small sails upon which the current of air acts like the air upon the sails of a wind mill. The number of revolutions which are made are registered by gearing in connection with a counter. A delicate form of this instrument is made by *Casella*. In France and Belgium an instrument constructed on a similar principle is used. It is known from its originator as *Combes' anemometer*. These and other instruments chiefly differ from each other in the number and nature of their sails, the great object being to obtain one which, whilst being self registering, is sufficiently frictionless and delicate to move in a light current. By the use of an empirical formula the velocity of the current of air is deduced from the number of revolutions indicated by the counter.

In the second class of instruments we have *Dickinson's anemometer*. This consists of a small rectangular frame swinging on an axis. From the number of degrees it is deflected when held at right angles to the current, the velocity of the air is determined.

The velocity of air in a level may also be measured by observing how long some light material like smoke takes to travel over a given distance. This smoke may be produced by the explosion of a small quantity of gunpowder. First the time of the explosion is noted and subsequently the time at which the smoke arrives at some given point.

From these observations the velocity is readily deduced. In making such observations the two points of observation ought to be in a straight part of the level so that the person who records the arrival of the smoke may have been able to note the time at which the explosion took place. The section of the level ought also to be regular or there will be variable velocities at different points along its length. Also it may be observed that varying results will be obtained according as we note down the arrival or the final disappearance of the smoke.

A still rougher method of approximately finding the velocity of air in a level, is to walk along the level in the direction of the current carrying a lighted lamp or candle. If you move slower than the current, the flame is blown and inclined before you; if you walk faster, it is blown back towards you; whilst if you walk at the same rate it will remain vertical, and your rate of walking measures the velocity of the current. In such determinations as these it is well to take a mean of several observations.

The velocity of air will vary at different points in the same section of a level. It therefore must be measured at several places and a mean velocity calculated. When once calculated for a given section, as the ratio of the mean velocity to the velocity at any point in the section is constant, subsequent measurements may be made at one selected point in the section.

The Equivalent Orifice (*Murgue*).—

Q = quantity of air in cubic feet per second passing through the opening (*i.e.* circulating round the mine).

h_a = ventilating pressure in feet of air column required to overcome the resistance of the mine.

A = opening in a thin plate in square feet (*i.e.* equivalent orifice).

K = coefficient of contraction of orifice (*i.e.* Vena contracta = .65).

Then $$Q = \sqrt{2gh_a} \times .65\ A$$

$$\therefore\ A = \frac{Q}{.65\sqrt{2gh_a}}$$

If Q is in thousands of cubic feet per minute and *ha* in inches of water gauge,—

$$A = \frac{.37\ Q}{\sqrt{h_a}} \quad \therefore \quad Q = \frac{A\sqrt{h_a}}{.37} \quad \text{and} \quad h_a = .1369\left(\frac{Q}{A}\right)^2$$

The average value of A for English mines is about 20 and for Belgian about 8.6 square feet.

Obstructions to the flow of Air.—In practice it will be found that the velocity of air calculated from the *head*, will be greater than the velocity actually observed by experiment. This is due to having omitted to consider the various resistances, which the air meets with as it travels along the galleries of a mine. These resistances are those of friction. Obstructions and contractions, usually referred to as stopping the flow by arresting its momentum, are really to be considered as producing large amounts of internal friction in the flowing mass of air.

When one body moves or slides across another, there is a certain resistance to the motion which we call friction. The *amount of this resistance* depends upon the *pressure* which these two bodies exert upon each other. If we could imagine that a force due to the head of air might be employed in sliding a body along a level, the resistance to the motion of such a body would be proportional to the weight or to the pressure which that body exerted on the bed over which it moved. This would be solid friction.

In the case of air moving along a level under the influence of the pressure caused by the *head*, although we know that the pressure is transmitted by the air in the levels to the surfaces over which it moves, the resistance is not, as in the case of solid bodies, proportional to the pressure. We have here a case of fluid friction, which is *independent of pressure*, being as great for a small pressure as for a large.

Some of the laws governing fluid motion are however evident. First it will be observed that the resistance to the flow of a column of air will depend upon the *extent of surface* which bounds its flow.

To take advantage of this, air ways ought to be made as *short* as possible, because a long level will offer the greater resistance to the passage of air. As the rubbing surface in a level depends upon its *sectional perimeter*, for a given sectional area we ought to make this perimeter as small as possible. Thus a level of circular perimeter may have a larger area and therefore allow a larger quantity of air to flow along it, than a level of rectangular section with an equal perimeter.

Therefore a circular level so far as the friction of air is concerned, would be the best form for an air passage. Lastly, the sides of an air passage ought to be made smooth, because they present less rubbing surface than one which is rough.

Not only is the resistance to fluid motion dependent on surface, which in a mine is measured by the length and perimeters of its levels, but it is also greatly dependent upon its *velocity*.

As the air travels along a level we may imagine it arranged in rings, those which are nearer to the sides of the level being retarded more than the air in the centre.

From experiments it is found that for small velocities, the frictional resistance is proportional to the velocity, for greater velocities it is proportional to the square, cube, and higher powers of the velocity. With the motion of water in pipes, conditions similar to those which would be obtained for air moving very fast, are found to exist.

In the following formulæ, which give the head of air necessary to overcome friction, we suppose the friction to be proportional to the square of the velocity, because this is found to agree well enough with experiments upon the flow of air in mines, but it must be remembered that this law is only roughly true.

Let F = force of friction in lbs.
k = is a number found by experiment.
l = length of openings traversed by air.
c = the mean perimeter of cross section of these openings.
v = velocity of air in feet per second.
Q = quantity of air in cubic feet per second.
A = area of cross section in square feet.

Then $F = k.\ l.\ c.\ v^2$, also $v = \left(\frac{Q}{A}\right)$

$$\therefore \quad F = k.\ l.\ c. \left(\frac{Q^2}{A^2}\right)$$

Now if the friction is overcome by a *head h* it is overcome by an urging force $h.\ A.\ w$, (where w is the weight of a cubic foot of air,) which is the equivalent of this *head*,

Therefore $$h.\ A.\ w = k.\ l.\ c. \left(\frac{Q^2}{A^2}\right)$$

$$h = \frac{k}{w} \cdot \frac{l\,c\,Q^2}{A^3}$$

Now $\frac{k}{w}$ is a number determined by experiments in mines where obstructions due to timbering &c. render the conditions somewhat different to those obtained for the flow along an even channel. André gives for this constant .00066.

Augmentation of a Natural Current.—All means taken to augment a natural ventilation which involve additions to the ordinary arrangements of the mine, may be considered as being, in a greater or less degree forms of artificial ventilation. Just as in all rooms to which there is an opening, there is more or less circulation in the air, so it is in all mines, and as by the opening or shutting of certain doors we are able to increase the draft in a room, so the draft which exists in a mine may be also increased.

Towers.—From what has already been said about the natural ventilation of a mine, it will be seen that if the temperature in a mine be different from that outside, the greater the difference in level between the opening at which the air enters and that from which it leaves, the greater will be the ventilation current. Where this difference in level does not exist, it may sometimes be conveniently created by building a chimney over one of the openings.

As the quantity of air passing out of any shaft is directly proportional to its velocity, we see that if the temperature producing motion remains constant, the quantity must he proportional to the square root of the height (see page 177).

We have therefore the velocity and quantity of air varying as $\sqrt{H}$. If we increase the length of the shaft by a tower h, the velocity and quantity of air will be proportional to $\sqrt{H+h}$. And if we compare the ratio of increase in a shaft H, and in a shaft H surmounted by a tower h, this increment will be measured by

$$\frac{\sqrt{H+h} - \sqrt{H}}{\sqrt{H}}.$$

If *H* is very large, that is if a shaft be deep, it will be seen that the above ratio, or ratio of augmentation, for a given chimney *h*, becomes very small, a result which might naturally be expected.

This difference in level, in hilly districts, may often be formed by driving special galleries.

Winds.—In the tropics where for long periods there are constant and steady winds, the pressure which these exert may be taken advantage of, to force a current of air through the workings of a mine, by the opening or closing of suitable entrances or levels.

Cowls.—The action of the wind may also be taken advantage of by placing over the *upcast* and *downcast*, or over openings connected with these, towers terminated by moveable cowls. The cowl which is over the *downcast* is so arranged that it always presents its open face to the influence of the wind, in consequence of which a current of air will always tend to pass down into the mine. The cowl over the *upcast* is always in a reverse position, so that the wind as its rushes past its edge, tends to drag the air from the interior along with it.

Pipe and Cap Head.—This is a pipe with a moveable mouth-piece placed at right angles to its length. This latter portion is above ground and is kept facing the wind, whilst the pipe extends down to the workings. The action is similar to the cowls on air towers, or to the ventilating arrangements we see on board ships.

Air Sollars.—An *air sollar* is formed in a level by carrying the tramway on a board floor raised about 6 inches above the actual floor of the level. This floor which divides the level in to two unequal portions may be made air tight by covering the joints with clay. If the floor of the level is not too steep, and there is not a rapid current of water flowing beneath it, a current of cold air will set up the *air sollar* beneath the floor, whilst heated air will pass out above the floor, along the level.

Air Head.—In the Staffordshire coal districts it is sometimes customary for ventilative purposes to drive a small drift or *air head* above a main level. With this it is connected by vertical openings called *spouts*. The bad air from the main level ascend by these *spouts* into the *air head* which leads it away towards an *upcast*.

Falling Water.—In case of an emergency water may be allowed to fall into the *downcast*. This by cooling and also by the draging action it exerts upon the air in the shaft tends to create a current. In the event of an explosion or the breaking of a machine, by which the ventilative currents had been deranged the turning of a properly arranged water cock might be used to restore the ventilation.

By Cooling the Air.—We see that just as air may be caused to ascend by its being heated, so it may be caused to descend by having been cooled. Apparatus for assisting in the cooling and the ventilating of a mine have been designed on this principle. The general character of these machines is that of a large *still*, where air which is the material to be cooled, instead of flowing through the *worm*, flows downwards through the vessel in which this is contained. Through the worm an upwards current of water is kept in circulation.

Water Trompe.—This contrivance consists of a pipe usually from 9 to 12 feet in length down which water falls into a closed reservoir. In the upper part of this pipe there are lateral openings pointing inwards and downwards. As the water falls past these openings, it drags and sucks air from the exterior. On the upperside of the reservoir there is a pipe to lead away the air which is being continually entrapped. The water is kept at a level which covers the orifice by which it escapes, and thus prevents the simultaneous escape of the air.

The upper end of the pipe at which the water enters is slightly conical. Exactly beneath the pipe in the reservoir, there is a board placed above the opening at which the water escapes. This is struck by the falling water which being broken up, liberates the air.

The mean average effect for such machines is given at 12½ per cent, from which it will be seen that they could not be used unless water power was plentiful. Another obvious condition for the employment of such a machine is, that there should be every facility for draining away the water from the level down to which it falls.

Furnaces and Steam Jets.—It is evident that if we increase the temperature of air in an *upcast* we shall increase the current. For this purpose *jets of steam* and *furnaces* are used. *Furnaces* are placed sometimes at the top of a shaft, sometimes in the middle, but more

generally at the bottom, where without discussing the question it is evident that they would be in the most favourable position to produce the best effect.

Furnaces in Shafts.—During sinking processes, or for some temporary expedient, a fire held in an iron grate is sometimes suspended in a shaft for the purpose of heating the air and producing an upward current. Where an opening is narrow, and where there is much timber the danger of such a proceeding is obvious.

Furnaces above Ground.—Sometimes the draft in an *upcast* may be improved by making a connection between it and the chimney of some of the boilers or furnaces which may be near the mouth of the shaft.

When furnaces are placed on the surface of the ground their action is similar to that which we should obtain by placing the mouth of the shaft in communication with a chimney.

Furnace Ventilation may require 35 lbs of coal per *H.P.* per hour which is not economical.

Ventilation of this description may be useful in cases where the workings are not deep, or in cases where we have a natural ventilation which it is occasionally necessary to revive.

Furnaces under Ground.—Furnaces above ground are but seldom used, whilst furnaces beneath the surface are very common, especially at some of the English Coal mines. Where it so happens that a special shaft can be devoted to them they may be placed immediately beneath it. It is more usual however to place them 80 or 100 feet distant, connecting them with the shaft by means of an inclined level or *dumb drift*. This ought to be of sufficient length to prevent sparks reaching the *upcast*. In this way annoyance to the *onsetters* is avoided.

The smoke which comes to the top of the shaft may be carried away through an inclined side drift entering the shaft beneath the *settle*, and leading towards a chimney.

These furnaces are generally so arranged, that in case it would be dangerous to allow the air which is being drawn out of the mine to come in contact with the fire, they can be supplied with a special branch of fresh air led to them direct from the *downcast*.

Above they are covered by an arch. In order to safely protect the roof from fire, this arching is sometimes made double. So that the proper quantity of air shall pass beneath the fire bars, the space above them may be regulated by a sliding door. On either side of the furnace, arched air-ways are sometimes built. At the back of the furnace there is a chamber to catch flying sparks. The average dimension of a furnace is about 6 feet in breadth, by 8 or 9 feet in length.

At Monkwearmonth a furnace 12 feet wide with 3 feet above and 3 feet below the bars, gave 120,000 cubic feet per minute.

Sometimes several furnaces may be required. These may be either arranged in juxtaposition, or be placed separately and worked according to the requirements of the mine.

Over a shaft which is ventilated by a furnace, a tower may be erected. In the shaft, such supports as from their position if made of wood would be in danger of fire, must be replaced by iron, or else by stone or brickwork. The latter is preferable, first because the conduction of heat in brick work is less than it is with metal, and secondly because of the heat and the gases resulting from combustion, metal-work is slowly destroyed.

To supply the furnace with air taken directly from the *downcast* naturally weakens the current which has to pass through the workings, and as it may happen that only a certain district of a mine produces dangerous gases, these alone are prevented from coming in contact with the furnace, and some portion of the air which has passed through other portions of the mine may be used to feed the furnace.

In N. France at Anzin a small shaft is sometimes sunk near to the *upcast* for the purpose of supplying a furnace with air.

Work done by Furnaces.—As at different mines the conditions are usually so varied,—the nature and size of the mine, the quantity of the coal which is used, the part played by natural ventilation &c., it is difficult to compare the effects produced by various furnaces. To shew what has been done, it may be mentioned that at Hetton Colliery with three furnaces, 190,000 cubic feet of air, and with four furnaces as much as 220,000 cubic feet, have been drawn out of the mine per minute. At the Haswell Colliery the velocity of the air in the *upcast* has been 29 feet per second. And at Wallsend in the levels underground, a velocity of 21 feet per second has been

obtained by the aid of furnaces. The greatest heat in the *upcast* has been from 145 to 180° *F.*

In the north of England it is considered that a furnace 6 feet broad, and 6 to 8 feet long, using in 24 hours about 2 tons of coal will cause a flow of from 40,000 to 50,000 cubic feet of air per minute, and the water guage will shew a pressure varying from 26mm. to 42mm.

The average temperature of the air in the *upcast*, from a well arranged furnace, ought to be from 150° to 160° *F.*

Theoretical Considerations.—From page 178, we see that the velocity and quantity of air passing through the *upcast* is proportional to $\sqrt{T-t}$, that is v, varies as the square root of the difference of temperature in the *upcast* and *downcast*. If we call this difference d, and the quantity of air in cubic feet per second Q

then v and $Q \propto \sqrt{d}$.

The cost of the materials which produce this heat depends upon the increase in temperature they produce or d, and the quantity of air which passes on $\sqrt{d}$

therefore the cost $\propto \sqrt{d}.\ d.$ or $d^{\frac{3}{2}}$ or v^3.

Furnace Paradox.—If we take a series of examples we shall see that by doubling or trebling the temperature, we neither double nor treble the velocity nor the quantity of air which passes.

The first three columns of the following table illustrative of this result, are given by Burat.

Temp: of ascending current.	Velocity of current.	Nos. porpotional to quantity of fuel used.
30	43	86
40	51	153
50	58	232
60	64	320
100	79	711

If we suppose the above numbers given by Burant to have been obtained by experiment, then because the quantity of coal consumed $= c.\ v^3$

$$c = \frac{86}{43^3} = \frac{153}{57^3} = \frac{232}{58^3} = \frac{320}{84^3} = \frac{711}{79^3}$$

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Taking the mean of these equations, $c = .00121$.

For any given velocity we can now calculate a number proportional to the quantity of coal which will be used. In the following table the same velocities as those given by Burant are taken, and it will be seen that the quantities of coal which are burnt, accord pretty nearly with the quantities before given.

Velocity of current.	Nos. proportional to fuel used.
43	96
51	160
58	236
67	317
79	596

If we make our calculations simply from the theoretical considerations given on page 178, because v varies as $\sqrt{T-t}$ and the quantity of coal varies as v^3, we shall see that if we have the difference in temperature in an *upcast* and a *downcast* represented by the first column of figures in the following table, the second and third columns will respectively represent the corresponding velocities which will be obtained, and the quanties of coal which will be used.

Differences in temperature. or $T-t$.	Nos. proportional to velocity of current.	Nos. proportional to coal which will be used.
25	5	125
36	6	216
49	7	343
64	8	512
81	9	929
100	10	1000

From these considerations it will be seen that high temperatures besides rendering it impossible to use the *upcast*, necessitate the expenditure of enormous quantities of fuel as compared with that required to produce low temperatures or low velocities.

Employment of Steam.—Very often, as for instance at some of the collieries of S. Lancashire, the waste steam from an engine is allowed to escape at some point in the *upcast*. In this way the air in the shaft is heated and an upward current is produced. If the steam which is employed for under ground engines is supplied to them by

pipes from the surface, the radiation from these pipes also tends to accelerate an upward draft. At Wigan several experiments have been made to employ steam not only to create a draft by heating the air, but to act mechanically by escaping under pressure in the direction of the moving current.

These were carried out by allowing 14 or 18 small jets of high pressure steam to escape in the *upcast*. So that these should not interfere with each other, a cylinder about 1 foot in diameter and 6 feet long was placed above each. The remaining space not occupied by these cylinders was carefully closed.

One of the best results was the exhaustion of air at the rate of 6,250 cubic feet by the burning of 1 lb. of coal. In the same shaft the burning of the same quantity of coal produced an exhaust of 10,650 cubic feet per minute.

From these experiments it seems that it would not be economical to generate steam specially for ventilative purposes, but if steam is at hand, as for instance from the escape of an under ground engine, it may be used with advantage.

Ventilating Pressure &c. (*Merivale*).—

If M = motive column (Ventilating pressure) in feet of air of temperature in upcast.
D = depth of upcast in feet.
t, T, $t^{\backprime}$ = temperature of air in downcast, upcast and returns respectively.
P = ventilating pressure in lbs per square foot.
I = height of barometer in inches.
W = weight of one cubic foot of the return air in lbs.
Q = cubic feet of air per minute in main return.
X = lbs of coal burnt per hour.
Y = area of furnace in square feet.

Then $$M = D \times \frac{T - t}{459 + t}$$

$$T = \left\{ \frac{1.3253 \times I \times D}{\frac{1.3253 \times I \times D}{459 + t} - \frac{1}{5}P} \right\} - 459.$$

$$X = \left\{ \frac{W Q \times T - t' \times .238 \times 60}{14000} \right\} \times \text{say } 2.$$

$$Y = \frac{X}{10} \text{ roughly.}$$

$$P = \left\{ \frac{1.3253 \times I}{459 + t} - \frac{1.3253 \times I}{459 + T} \right\} \times D.$$

Theoretically about 150 cubic feet of air are required per lb. of coal burnt, practically 300 cubic feet.

Consumption of Fuel on Pit Furnaces (*Trans NEI., VI*).—

Locality.	Depth in	Coals per *H P* per hour excluding power due to heat of mine.	Coals per *H P* per hour including power due to heat of mine.	Quantity of air per mine.
	feet.	lbs.	lbs.	cubic feet.
Thornley 5/4 seam	556	85.5	37.5	45,756
Thornley Hutton seam.	868	162.4	57.1	26,574
Walker	960	30.5	15.6	44,800
Castle Eden	1038	29.1	28.3	42,326
South Hellon	1212	27.2	15.5	132,895
Wearmonth	1800	69.5	7.9	70,500
Average	—	60.7	27.6	—

Ventilating Machines.—The various machines which are used for purposes of ventilating mines may be divided into two great classes. First there are those machines working backwards and forwards with the motion of an ordinary pump, and secondly those machines which have a rotatory motion like a winnowing machine or fan.

Duck Pump (*Harzer Wettersatz*).—This consists of a cylindrical vessel closed at one end inverted over the end of an air pipe coming up through the bottom of a second vessel partly filled with water. On the closed end of the inverted vessel, and over the open end of the air pipe, there are valves, both of which open upwards. By raising and lowering the upper vessel it will be seen that air will be drawn up through the air pipe to be discharged through the valve into the inverted cylinder.

This up and down motion is often given by attaching the upper vessel to the pump rods.

By reversing the motion of the valves, instead of exhausting the air from the workings, air might be forced in.

By placing two air pipes with valves moving in opposite directions beneath a moving cylinder which is without a valve, air would be drawn up one of these pipes and forced down another. In this way bad air drawn from one point in a mine, instead of being discharged near to the pnmp, might be discharged some distance away. Small machines are 1½ to 2 feet in diameter.

Marihaye's Ventilator.—This ventilator is a more perfect form of the Duck pump. If we imagine the air pipe of the Duck pump to be increased until it almost fills the interior of the fixed cylinder, so that the movable cylinder has only a small annular space in which to move, we shall see the general arrangement of Marihaye's ventilator. On the top of the movable cylinder, which has a diameter of 3.66 met. and a height of 2.6 met. there are 16 valves.

The same number of valves are placed on the end of the cylinder which corresponds to the air pipe. The whole is constructed of sheet iron. Two of these may be worked by one engine, which is placed between them.

The useful effect is given as being from 33 to 39 per cent.

Struve's Ventilator.—Struve's Ventilator consists of a large chamber closed above and below, in the sides of which there are a number of valves. On one side communicating with the shaft these valves open inwards. On the opposite side they open outwards. Inside this chamber these is a large bell like the movable cylinder of a Duck pump working up and done in a chamber filled with water. When this rises, the air in the upper part of chamber is compressed and forced out of the valves which are on the opposite side to the shaft, at the same time the air in the lower portion expands, and the lower valves on the side of the shaft are opened whilst those opposite are closed. When the piston descends the reverse happens.

Another form of this machine is where instead of the water chamber, a cylinder within which there is a piston, is placed inside the exterior chamber. This cylinder is held is position by supports

from above and below. To divide the upper valves from the lower ones, a horizontal partition is placed between the centre of the chamber, and the cylinder.

These machines are usually worked in pairs by means of one engine.

In this latter arrangement where there is a piston working in a cylinder, it will be observed that a considerable amount of friction has to be overcome which is not felt in those forms where instead of the piston, there is a bell working in water. An objection which has been raised against these machines is that any sudden change of motion, will disturb so large a body of air that its effect may be felt in distant places, and be the cause of dangerous results. It may also be mentioned that the valves are apt to get out of order. Notwithstanding these objections this machine has been successfully ussed at many collieries, especially in S. Wales. At the Risca colliery two pistons 18 feet in diameter and 6 feet stsoke with a small engine and 35 lbs of steam gave 43000 cubic feet of air per minute.

The pistons have a diameter varying from 12 to 22 feet, and a stroke from 6 to 8 feet.

They are capable of exhausting from 20,000 to 100,000 cubic feet of air per minute. One machine gave for every lb. of coal which was burnt, 5233 cubic feet of air, with an average manometric pressure of 35^{mm}.

Piston Machines.—These consist of a rectangular or more usually a cylindrical case made from wood or iron. At one end this is open, and at the other end it is covered by valves. Inside this cylinder there is a closely fitting piston also covered with valves. As such an arrangement will be intermittent in its action, two cylinders are so arranged that they work alternately. These may be placed vertically and worked by means of a beam or they may be placed horizontally. The efficiency of ventilators arranged on the former of these systems has varied from 20 to 40 per cent, and with the latter it is said that as much as 65 per cent has been obtained. One of the first of these piston machines was erected in 1828 by M. Brisco at Charleroi. In 1858 it was improved upon by M. Mahaux.

Mahaux's Ventilator.—Here two pistons are placed at opposite extremities of a piston rod. The valves are suspended

vertically, this being the position in which they offer the least resistance to motion. These are large. One of these machines with pistons 4.9 feet diameter, gave 45,000 cubic feet of air per minute.

Nixon's Ventilator.—This ventilator consists of a chamber, at the ends of which valves are arranged as in Struve's ventilator. Moving backwards and forwards in this chamber there is a large piston. To lessen the resistance this is supported on wheels. At Navigation Pit near Aberdare, the piston which had a stroke of 7 feet, measured 30 feet by 22. The valves, which for lightness are made of wood and leather, measure 16 by 24 inches, and are 672 in number. Theoretically such a ventilator at 9 strokes per minute will gives 166,000 cubic feet of air.

An objection to these machines is that there is much leakage, and also that owing to the piston not being able to make a full stroke, the ventilative effect is interfered with, by the fact that between the end of the cylinder and the piston there is always a large space.

Centrifugal Ventilators or Fans.—Fans which are used for the purpose of blowing furnaces move with a high velocity and produce a small current of air at a high pressure. Fans which are used for mines are required to produce large volumes of air at a comparatively low velocity. All these fans consist of a wheel provided with vanes enclosed in a case. This has two openings one central and another tangential. As the fan is revolved air enters at the former of these openings and is expelled at the latter. By connecting first one of these openings and then the other with a shaft, air might either be forced into a mine or drawn out. The course which may be pursued will depend upon the conditions of the mine. Occasionally air may be forced into a mine, but the usual plan is to ventilate by exhaustion.

Since early times fans with radial vanes centered round an axis and enclosed in a wooden case have been used and even now for temporary purposes, and in small workings they are employed with advantage. By building these fans of sheet iron, and by altering the form of the case, great improvements have been achieved, but the chief alterations have been with regard to the position and form of the vanes.

Ventilating pressure of a Fan (*Merivale*).

H = gross ventilating pressure (theoretical depression) in feet of air column, of density of flowing air.

u = tangential velocity of fan in feet per second.

Then theoretically $H = \frac{u^2}{g}$

But as $H = \frac{V^2}{2g}$ we may say that the theoretical depression is double the height due to the tangential velocity.

With H as above

P = ventilating pressure in lbs per square foot.

WG = water gauge in inches.

d = density of water = 1000.

d' = density of air = 1.2 approx. at ordinary pressure and temperature.

Then $P = 5.2\ WG.$

$$H = \frac{d\ WG}{d' \times 12} = \frac{1000\ WG}{1.2 \times 12} \quad \therefore \quad WG = \frac{1.2 \times 12\ H}{1000}$$

The following formulæ are from Mr. A. L. Steavenson's translation of M. Murgue's work.

Let H = theoretical depression in feet of air column given by a perfect fan if its eye were shut off from the mine and atmosphere.

h_a = effective depression in feet of air column; *i.e.* the ventilating pressure required to overcome the resistance of the workings = the water gauge in the fan drift.

h_o = useless depression in feet of air column; *i.e.* the ventilating pressure required to overcome the resistance the air meets with in passing through the fan. To obtain this the communication between the fan eye and the workings must be closed and the eye connected direct with the atmosphere.

Q = the quantity of air in cubic feet per second.

A = Equivalent orifice in square feet.

O = Orifice of passage in square feet; *i.e.* the area of a hole in a thin plate which would offer the same resistance to the air that the fan offers.

k = coefficent of efficiency of fan (varying in the case of well designed fans from .5 to .8).

g = gravity, say 32.19.

u = tangential speed of fan in feet per second.

Then $H = h_a + h_o$

$$H = \frac{u^2}{g}$$

$$A = \frac{Q}{.65\sqrt{2gh_a}}$$

$$O = \frac{Q}{.65\sqrt{2gh_o}}$$

$$h_a = \frac{Q^2}{2g(.65\,A)^2}$$

$$h_o = \frac{Q^2}{2g(.65\,O)^2}$$

$$Q = .65\,A\sqrt{2g\,h_a}$$

$$Q = .65\,O\sqrt{2g\,h_o}$$

$$\frac{h_o}{h^a} = \frac{A^2}{O_2}$$

$$h_a = \frac{H}{1 + \frac{A^2}{O^2}}$$

$$Q = \frac{.65\,A\sqrt{2\,g\,H}}{\sqrt{1 + \frac{A^2}{O^2}}}$$

And for practical calculations:—

$$h_a = \frac{k\,u^2}{g\left(1 + \frac{A^2}{O^2}\right)}$$

$$Q = \frac{.65\sqrt{2\,k\,A\,u}}{\sqrt{1 + \frac{A^2}{O^2}}}$$

Combes and Redtenbacher say, that the vanes ought to be tangential to the exterior circumference and make an angle at the centre. Others whilst making them tangential to the exterior make them radial on the interior.

It seems to us that they ought to make an angle at the circumference and also at the centre, and these angles may be calculated from known principles.

However it is not of any importance what is the shape of the vanes of a fan, except that from considerations of friction, they ought to be made as direct as possible. The angle made by the vane at its inner edge with the direction of motion must be such that the air enters the vanes without shock. That is, if A is the area in feet of all the openings round the inner edges of the vanes, Q the quantity of fluid flowing per minute, n the number of revolutions per minute, r the internal radius, and θ the angle which the vanes make with a tangent to the inner circumference, then,

$$\tan\,\theta = \frac{Q}{2\,\pi\,r\,n\,A}$$

The angle made by the vanes with the outer circumference may have almost any value, if there is a large enough space surrounding the fan. The discovery of the fact that centrifugal fans have nearly all the same efficiency, if they are all surrounded with a large enough space in which a natural whirlpool can establish itself, is due to Professor James Thomson.

Eckardt's Ventilator.—This ventilator has six vanes, each of which is bent to form a semicircle, the radius of which is equal to quarter the diameter of the whole wheel.

Diameter of the wheel...	16 inches.
Breadth	4 inches.
Exit epening	4 square inches.

The whole is made of sheet iron and and is driven by means of a band, at the rate of 500 to 1000 revolutions per minute. The useful effect is 20 per cent.

Two such wheels may be placed on the same axle, being so arranged by suitable divisions, that one will drive fresh air into a working whilst the other sucks out the bad air, and drives it on to some distant point.

Reichenbach and Goylays's Ventilator.—The peculiarity of this ventilator is that the metal vanes are replaced by a circular brush made from wire or whale bone. It is probable that the effeciency of this machine is low.

Rittinger's Ventilator.—In this ventilator the wings are so constructed that the curvature of the vanes on the outside is tangential to a radius, and on the inside they are tangential to a line which makes an angle of 47° with a second radius at the point where they meet the inner circle. The whole is enclosed in a case. It makes from 300 to 600 revolutions per minute.

A ventilator with a radius of 19.6 inches and 15 vanes, gave 52.37 per cent of the calculated work.

This ventilator is also constructed on a large scale. One of 13 feet diameter with vanes making 80 to 400 revolutions per minute extracted about 400 cubic feet of air per second = 24,000 per minute.

Ventilators for a whole mine.—These always act by exhaustion and therefore with but few exceptions they can be used without being enclosed in a case. They are generally placed at some distance from the shaft, with which they communicate by means of an inclined tunnel.

Nasmyth's Ventilator.—At Abercarn in S. Wales, one of these ventilators had 8 radial wings formed from sheet iron, each 3 feet long and 3½ feet broad. The total diameter of the machine was 13½ feet. The foul air coming from the mine passed up between two side walls and entered at the centre of the fan to be discharged at its circumference.

It was driven by an Engine of 13 Horse power and made from 60 to 80 turns per minute.

The work done per minute was,—

By natural ventilation	26,545	cubic feet.
„ 60 revolutions.....	49,314	„ „
„ 80 revolutions.....	61,704	„ „

Letoret's Ventilator.—This ventilator usually consists of four flat wings which are inclined at angles of from 105° to 135° at the extremity of four radial arms.

The diameter of these machines is from 6 to 10 feet. They make from 200 to 300 revolutions per minute, and the useful effect is from 23 to 26 per cent. Burat gives in two cases a usuful effect of from 40 to 44 per cent.

If the inlet is too large, it is said that two draughts may arise, one entering the machine near the axis and the other flowing out. The quantity of air is proportional to the velocity, the manometric pressure to the square of the velocity, and the power of sucking to the cube of the velocity.

If a high pressure is not required, this machine recommends itself on account of its simplicity and cheapness. It is much used near Mons. It gives a gauge of 2 inches and over.

Guibal's Fan.—Amongst contrifugal ventilators, this is perhaps one of the commonest which are employed at mines. It consists of from 4 to 10 vanes braced in such a way to a series of radial spokes, that both strength and lightness are obtained. These vanes are made from sheet iron, and are slightly bent forwards. The opening by which the air is received from the shaft is large. The one by which it is discharged occupies about one quarter of the circumference, but its size can be regulated by a movable shutter. Outside this latter opening there is a chimney, which from its base to its summit increases in its dimensions. In this way, before the discharged air comes in contact with the outside air, its velocity is gradually reduced.

A large ventilator of this description at Usworth Colliery near Newcastle had the following dimensions.

Diameter	45 feet.
Breadth	12 feet.
Number of wings...	10.

With 40 revolutions per minute it extracted 147,000 cubic feet of air, with a pressure of 6mm.

The efficiency of these ventilators is given by Lottner and Serlo been 44 and 64 per cent.

By adding a second of these expanding chimneys, but placed in a reversed position and connected with the shaft, air may be either drawn out or forced into the mine.

These ventilators are usually from 20 to 30 feet in diameter and make from 50 to 100 revolutions per minute. The smaller number of revolutions is preferable.

Guibal Fan experiments at Staveley gave the following results.

1. The quantity of air discharged is proportional to speed of fan.
2. The water gauge increases proportionally to the square of the number of revolutions of the fan per minute.
3. The *H.P.* in the air increases as the cube of the quantity.
4. The steam pressure in the cylinder is in proportion to the square of the piston speed.
5. The *H.P.* of the engine is proportional to the cube of the number of revolutions per minute, or to cube of volume of air (R. Howe. Trans: Chesterfield &c. Institute of Engineers Vol. I. p. 55).

The useful effect ranges from 56 to 70 per cent of the steam power of the engine and the consumption of coal from 7 to 16 lbs per *H. P.* per hour.

Duty of Fan at a Fair Speed.

Diameter feet.	Width feet	inches.	Volume of air per minute cubic feet.	Water Gauge in inches.	Stroke inches.	Diam of cylinder inches.
10	4	0	20,000	0.50	6	12
12	4	0	30,000	0.65	12	12
16	5	6	40,000	0.80	12	18
20	6	6	50,000	1.20	18	18
24	8	0	70,000	2.00	20	20
30	10	0	100,000	2.75	24	24
36	19	0	150,000	3.50	30	33
40	12	0	200,000	4.25	36	36

The duty varies with the condition of the shaft, air ways &c.

Waddle Fan.—This fan discharges its air round the circumference into the atmosphere. The air passage through the fan has a decreasing section from the centre to the circumference, so that the velocity of revolution at any distance from the centre, multiplied by the sectional area of the passage at that distance, is preserved constant with a view of keeping the fan filled up to its circumference with issuing air, and preventing the possibility of reentries of air (Oliver & Co. Catalogue).

The blades which are inclined backwards and the casing, are in one piece and revolve together.

The engine is geared on one side while the air enters on the other.

The Schiele Fan. — This is a high speed Ventilator usually driven by a belt, receiving air on both sides and discharging into a volute formed air chamber increasing in area round the tips of the blades. The blades are broad in the center and taper to the tip. They revolve between walls following their form. It has an expanding chimney.

The Capell Fan.—This fan consists of a revolving cylinder in which port holes are cut. The wings are attached from the edges of the port holes on the inside and the outside of the cylinder. The discharge is to an expanding chimney. The port hole area is greater than that of the inlet. The fan is small, being from 8 to 15 feet in diameter, and is run at a high speed. It is stated to have given a water gauge of 10 inches and reached an efficiency of 80 per cent. It is driven by belt or rope gearing making from 100 to 300 revolutions per minute.

Other modern fans are the Bowlker, the Medium, and the Rateau.

It must be remembered that the results obtained from a fan at one colliery will be different for the same fan at another colliery—the conditions at the two mines being different. The Austrian Fire Damp Commission recommends large fans with low speed where there are large air ways, and small fans with a high speed where the air ways are narrow.

Brunton's Ventilator.—This is a ventilator where the axis of the fan is placed vertically.

Audemar's Ventilator.—This ventilator is somewhat similar to Guibal's but like Brunton's it turns upon a vertical axis. The air from the mine is drawn in at the centre and is discharged round the circumference. The air which is entering is separated from that which is leaving by a rim of metal on the under side of the wheel which dips into a circular trough filled with water built on the top of a wall surrounding the axis.

An objection to this machine is that there is a large weight supported on the extremity of an axis, in consequence of which it is constantly requiring to be repaired.

Harzé's Ventilator.—This also is a ventilator which works upon a vertical axis. It consists of a large number of curved vanes. Round the circumference of this there are a large number of fixed vanes between which the air has to pass before it can escape. Each pair of these, acts like Guibal's expanding chimney in gradually reducing the velocity of the out-going air.

Combe's Ventilator.—This is of historical interest. Although not intended, it reduces the velocity of the outgoing air to a minimum. At first it was constructed with a vertical axis, afterwards with an axis which was horizontal. Also at first it had only one inlet for the passage of air, but later on, it was constructed with two. Its efficiency is given by different writers as being from 15 to 39 per cent.

Screw Ventilators.—The action of these may be regarded as the reverse of a wind mill. The following are examples of this type of ventilator.

Lesonine's Ventilator.—The efficiency of this machine is given as being from 25 to 26 per cent.

La Motte's Ventilator.—This consists of a horizontal cast iron cylinder on which there is a sheet iron screw. It is driven by an endless band. The maximum efficiency is given as being about 20 or 21 per cent.

Guibal's Hydropneumatic Ventilator.—This was a screw made out of sheet iron about 16 feet in diameter, placed horizontally and partially immersed in water.

Theoretically it will give extremely good results, but so far, Burat considers that this form of ventilator has not yet been fairly tested.

Pasquet's Ventilator.—The efficiency of this ventilator is given 27½ per cent.

Ventilation Wheels.—Under this class of ventilators we include those contrivances which act by entrapping the air, and then as they revolve sweeping it out before them.

Fabry's Ventilator.—The action of this ventilator is similar to that which we should obtain by the revolution of two wheels furnished with long projecting teeth so placed that as they revolve, the ends of the teeth come in contact and sweep along with each of them a certain quantity of air.

As constructed Fabry's ventilator consists of two revolving axes on which are fixed three radial vanes. On these, cross vanes are placed. The edges of these cross vanes have epicycloidal curvatures. The two axes are placed at such a distance, that when they revolve in opposite directions, the epicycloidal surfaces, which may be covered with felt or leather, roll over each other.

The whole of the arrangement is placed in a case or trough, the sides of which are smooth and so curved that when the vanes revolve the ends of the radial spokes just pass along the walls.

When the vanes revolve towards each other, a certain portion of air is enclosed and carried downwards, but a larger portion is at the same time swept from below to the right and left upwards. The difference between these two quantities is the air which would be drawn out of the mine, which by the movement we have here supposed, would result in exhaustion of air from the mine.

This ventilator has been constructed with eight vanes and also with only two.

The efficiency of a small ventilator of this description has been given at 51 per cent.

Lemielle's Ventilator (*Belgian*).—This consists of a cylinder and a drum, the latter revolving eccentrically upon a horizontally placed axis within the other.

On the internal drum, shutters are placed. By means of rods moving upon a fixed axis, at the point where the eccentric cylinder closes against the internal surface of the large cylinder, these shutters are drawn in, whilst in other portions of its revolution they open out.

The form of the internal drum will depend upon the number of shutters which are used. When there are three shutters, the drum will have three sides. In place of rods springs may be used.

One ventilator 10 feet long with 6 shutters each 4 feet, 3 inches long, with 16 revolutions, gave from 225 to 235 cubic feet of air per second, with a manometric depression of from 2 inches to 2¾ inches.

The efficiency is from 62 to 65 per cent.

Root's Blower.—This consists of two wooden wing wheels each of which has the form of a figure 8. These are fixed at right angles to each other on separate axes in such a mananer, that when they are caused to revolve in opposite directions, the convexity of one fills the concavity of the other.

These are covered by a case in such a way, that when they revolve, during half of every revolution, they glide over its semi-circular sides. Outside the case, the axes of these moving vanes are connected by two toothed wheels.

These are sometimes 6 or 7 feet long and 2 feet 9 inches broad. They are driven by a band at a rate of from 200 to 250 revolutions per minute. They are chiefly used for cupolas and forges.

Evrard's Ventilator.—This consists of two cylinders of unequal diameters. Along the smaller of these there are two epcycloidal grooves. Projecting from the larger, are four radial vanes so arranged that as the two cylinders, which are placed side by side, revolve, they sweep through the epicycloidal hollows. According to the direction in which these cylinders revolve, air will be either swept outwards or inwards.

Cook's Ventilator.—As in Lemielle's, there is a cylinder working excentrically in a cylindrical chamber.

To prevent the ingoing air meeting with that which is going out, a shutter is, by the pressure of the air and a spring, kept in contact with the movable cylinder. One of these at Bishop Auckland had a rotating cylinder 8 feet in diameter and 16 feet long.

General Notes on Ventilators.—Sketches and general descriptions of many of the above mentioned Ventilators will be found in Lottner and Serlo's Bergbaukunde. Very many of them are only of historical interest.

The fans in general use are the Guibal, Waddle, Schiele, and Capell. They are described by C. Pamley in his Colliery Managers Handbook.

Considerations respecting Natural Ventilation.—An objection to natural ventilation is that it is liable to variation and even to stoppage. One of the chief causes which tend to produce variability, is the change at various seasons in the temperature of the atmosphere.

For instance, it may happen that in summer time the external air is warmer than the air in the mine, whilst in winter time the reverse occurs. The consequence of this would be, that the force tending to produce a current, would during these different seasons act in different directions. And twice during every 12 months the external temperature being equal to that in the mine, these forces would balance each other.

As the differences in the strength of a ventilating current due to variations in external temperature will be different in different districts, in order to study say their monthly variations, a table shewing the mean monthly temperature must be constructed, and compared with the under-ground temperature of the mine which in most cases may be regarded as being nearly constant.

A curve passing through a series of ordinates, the lengths of which were proportional to the square root of the difference in temperature of the outside air and that in the mine, would give a graphical representation of the variation in the force of the ventilating current.

The fact that the heat which is received from the surface of the earth travels downwards in a series of slowly advancing waves, would up to a certain depth, tend to make the difference between the outside air and that in the mine, sometimes greater, and sometimes less. These effects are however too small to be of any practical importance.

Owing to the variability of a natural ventilating current, it is often necessary to supplement its action by means of a furnace or some machine.

Another objection to a system of natural ventilation is, that it is seldom possible to increase its effect above a certain point. In certain cases, as for instance in mines subject to sudden outbursts of gas which it is necessary to sweep away, such a defect would be serious.

Although natural ventilation may be subject to variability, it will seldom be subject to a sudden stoppage such as might be expected from the breakage of a machine.

If any of the devices which are sometimes resorted to in order to increase the natural ventilation, as for instance the building of a chimney, tend in any way to obstruct or narrow a passage, their use will not produce the effect which is expected from them. The same remark will also apply to furnaces and machines.

Considerations respecting the use of Furnaces.—An objection to the use of furnaces is, that in the case of fiery mines, unless special means are taken to supply them with fresh air, accidents may arise.

In the North of England, notwithstanding the fact that many of the mines are fiery, furnaces are used for ventilation. On the continent it is more usual to employ machines.

From the following table, Burat endeavours to shew, that the conditions under which the coal miners work in France, where fans are chiefly employed, must be better than the conditions under which the English miners are employed.

		France 1864.	England 1865.
1.	Production	11,242,634 tons.	100,493,748 tons.
2.	Number of workmen	76,666.	315,451.
3.	Annual production per wokman.	147 tons.	318 tons.
4.	Number of wokmen killed	186.	984.
5.	Proportion of killed to total number of workmen.	.0024.	.0031.
6.	Quantity of coal produced corresponding to each workman killed.	60,444 tons	102,127 tons.

From 3 we see that the French coal miner only does about half the work done by an English miner, and also that for the production of a given quantity of coal, double the number of Frenchmen are exposed to accident as would be exposed in England.

From 6 we see that for the extraction of a given quantity of coal there are only about half as many accidents as there are in France.

From 5. we see that as compared with the total number of persons, exposed to danger there are fewer accidents in France than in England.

It is on this that Burat lays stress and in consequence concludes that where fire damp occurs, furnaces ought to be replaced by fans.

Before accepting such conclusions it would be necessary to carefully examine the statistics and general conditions of several groups of mines.

In England although annually there are about 1000 death at the collieries not more that ¼ of them are caused by explosions, and many of these explosions are caused by other means than by furnaces.

For example during eleven years we find that the deaths in English coal mines were divided as follows,—

From explosions of fire damp	23.57	per cent.
From falls of roof and sides	40.77	„ „
From miscellaneous causes	35.66	„ „

In the case where a mine is ventilated by means of a furnace in order to double the quantity of air we are obtaining, the quantity of fuel must be increased many fold. Regarded in this way, ventilation by means of a furnace may not be economical.

This however can to a great extent be overcome by using a pair of furnaces, or else a furnace which is very long and of large area.

The action of a furnace will largely depend upon the wetness or dryness of a shaft, and the conductivity of its walls.

The first cost of a furnace is small, and in coal mines where fuel is cheap the cost of working may be also small.

Considerations respecting the use of Machines.—When machines are used for ventilating purposes, air may be either forced into a mine or sucked out. When the former method is persued there is a tendency to prevent the escape of gas, but when the latter is followed the barometric pressure is to some degree lowered, and the escape of gas is facilitated. In mines therefore which are subject to the escape of gases, the former method may perhaps be found to be the most advantageous. To meet all circumstances, many ventilators are contrived to work on either principle.

In the case of Guibal's fan when it is employed to force air into a mine, according to Burat it only gives ⅗ of the pressure it gives when used in sucking out. For this reason it is found preferable to draw the air out from a mine rather than to force it in.

Unlike furnaces, the first cost of a machine and subsequently keeping it in working order is great.

Because it appears that it is more economical to produce a draft by means of a fan than by burning fuel to heat a chimney, it would also seem that it must be more economical to create a current by means of a properly constructed ventilator.

The action of a machine does not depend so much upon the dryness of a shaft as is the case with a furnace, also by an explosion, a ventilator is not likely to be interfered with.

With a ventilator the current of air can be more easily varied than it can with a furnace.

Centrifugal ventilators are cheap in construction and are good where only a small pressure is required. If they are made to give a high pressure their efficiency diminishes. Fans of the Guibal type are the best.

When these are very large they are apt to be deranged by being driven at a high rate, and also they allow of considerable leakage.

In cases where large pressures are required, a piston ventilator constructed like an air pump may be used. As a rule, however, the pressure they produce is greater than that which is required in ordinary mines.

The chief objection to these machines lies in the working of the valves, there being great difficulty in balancing them, especially in cylinders which stand vertically.

In "bell machines" like the duck pump or Struves' ventilator, the same difficulty is experienced. In addition to these difficulties in working, it may be mentioned that the first cost is generally great.

Amongst the various wheel ventilators, Fabry's is perhaps the best.

The first cost of these machines is greater than that of the centrifugal ventilators. Unlike them however, they give the best results at high pressure.

To calculate the efficiency of a Ventilator.—The efficiency of a ventilator like that of any other machine, is the ratio of work given to it as compared with the work which it actually performs.

If a ventilator is driven by means of a steam engine, indicator diagrams will furnish us with the means of calculating the Horse power which is supplied to it. This we will call *HP*.

The work done by the ventilator will be equal to the volume o air which passes any point per minute, which we will call V, multiplied by the greatest difference in pressure which it exerts in lbs. per square foot, which we will call P, this difference ought to be that between the air in the mine before the machine is working, and that after the machine is in motion, taken in some open portion of the mine to avoid any effect from a current. The work done will therefore be represented by the expression VP.

This expressed as horse power will be $\frac{VP}{33000}$

The efficiency of the ventilator will therefore be,—

$$\frac{HP \times 33000}{VP}$$

Efficiency of a Furnace as Compared with a Machine.—Let w lbs. of coal be burnt per minute, and let the units of heat given out by 1 lb. coal = 5040. (Welsh coal), and let 1390 units of work be the equivalent for 1 unit of heat. (Joule's equivalent centigrade.)

Then $\frac{w \times 5040 \times 1390}{33000}$ = Horse power of such coals in a perfect engine.

Let e_1 = efficiency of engine, (or the ratio of the work done by the engine as compared with that which is applied to it).

Let e_2 = efficiency of a machine.

Then $w \times 5040 \times 1390\, e_1 e_2 = V.P.$ (1)

If e_3 = efficiency of a furnace as a draught producer then $w \times 5040 \times 1390 \times e_2 = V.P.$ (2)

Comparing e_3 with the product $e_1\, e_2$ we see when a furnace will be better than an engine and fan.

At the Hollingwood and Springwell Collieries it appears that the saving in fuel by fans compared with furnaces, was about 83 per cent. The calculations were made upon the quantity consumed per *H. P.* in the air (R. Howe. Trans. Chesterfield &c. Institute of Engineers Vol. I. p. 60).

The greater efficiency of a fan as compared with a furnace will vary according to a variety of local circumstances. The deeper the shaft the greater will be the efficiency of the furnace.

Relation between Volume Pressure &c. (*Merivale*).

The ventilating pressure varies as:—

The depth of the upcast (furnace ventilation).
The difference of temperature in upcast and downcast.
The *H.P.* of the ventilating machine or furnace.
The quantity of coals burnt.

The quantity of air circulating varies as:—

The revolutions of the fan.
The tangential velocity of the fan.
The square root of the ventilating pressure.
The square root of the depth of the upcast (furnace ventilation).
The square root of the difference of temperatures (nearly).
The cube root of the *H.P.* of the ventilating machine.
The cube root of the coals burnt.

The student may compare this summary with formulæ already given.

Pressure to produce a Current.—The amount of pressure required to drive a given quantity of air through a mine chiefly depends upon the amount of friction which has to be overcome. Because of the complication of this and other conditions it is necessary that the requisite pressure should be determined by experiment.

A ventilator ought to be capable of producing a pressure of from 1 to 4 inches of water. Where much fire damp is present which it is necessary to sweep away, pressures of from 2.5 to 3.5 inches may be necessary.

One inch of a water column represents a pressure of about 5.3 lbs. per square foot. These pressures might be calculated as being equivalent to a column of air so many feet in height.

Velocity of Air.—The velocity at which air passes through the workings of a mine is usually about 3 feet per second. If the velocity is much less than this it may not be sufficient to carry out the deleterious gases, whilst if it is 5 feet per second it will be sufficient to force burning gases through the wire gauze of a Davy lamp and thus be the cause of an explosion.

In the upcast the velocity sometimes reaches as much as 28 feet per second.

Quantity of Air Necessary for a Mine.—The quantity of air sent through a mine will depend upon the quantity of deleterious products which it is necessary to sweep out.

If the number of persons in a mine..... $= m$
„ „ „ „ horses ..... $= h$
„ „ „ „ pounds of powder used every hour $= p$
„ „ average out-put of coal in tons per minute..... $= q$
„ „ exposed surface of coal in square yards..... $= s$
and the ventilative current in cubic feet per minute ..... $= V$

Then André says,

$$V = m\,24 + h\,72 + p\,192 + q\,100 + s.$$

In the North of England 100 to 500 cubic feet of air per minute per man and 30 to 160 cubic per minute per ton of coal extracted per day is supplied. The velocity is about 4 feet per second.

Sometimes we see that the quantity of air passing through a mine is considered in proportion to the quantity of coal which is extracted. As explosive gases are chiefly given off from the surface of freshly broken coal, this method of regulating the ventilation of a mine will be valuable where such gases are largely developed.

In Belgium 60 to 120 cubic feet are allowed per minute for each miner. The Mining law of Pennsylvania requires 200 cubic feet per minute per man. Each horse will require three time as much as a man. At Mount Cenis 1 lb. of gunpowder was calculated to require 1616½ cubic feet of air, but 4043 cubic feet was considered to be a better allowance (see p. 163).

These quantities, especially those having reference to the quantity of coal which is extracted, are given differently by different authorities, and although they are sufficiently great for an ordinary mine, for the case of a mine giving off large quantities of deleterious gas it would be well to increase them.

In some large collieries from 150,000 to 300,000 cubic feet of air and even still larger quantities are caused to circulate per minute.

Admission of Air into a Mine.—A single shaft will often ventilate itself by a current of warm air tending to rise up the center

whilst the external air is pouring over the edges and creeping down its sides.

In a level, the cool air will tend to creep along the floor, whilst the heated air will flow out along the roof.

These actions will be better marked in winter than in summer, when the respective courses taken by the heated and cool air may be reversed.

For the ventilation of small mines it is often sufficient to have a single shaft divided by a brattice, one side of which is used as a "*downcast*" and the other side as an "*upcast*." If possible however it is better to have two shafts. In the English mines, the employment of two shafts is made a necessity by law, as is also the case in Pennsylvania.

In order to prevent any retardation in the flow of air, the cross section of the upcast is made about $\frac{1}{3}$ larger than the downcast. This is in order to allow for the expansion which the air has undergone by passing through the workings or the mine. The *return* air ways ought to be similarly increased.

Ventilating a Single Level.—The ordinary method of doing this is to carry a division or "*brattice*" down the centre of the level almost up the working face.

If the level to be ventilated has its exit in the open air, it may be necessary to force air down one side of the brattice by means of a machine.

Ventilation of Parallel Levels.—As the ventilation of a single level requires a brattice, in coal mines, levels or *bords* are driven in parallel pairs, and to allow the air to pass, they are joined by passages called *thirlings*. When the ends of these levels have advanced some distance beyond a *thirling*, a second *thirling* is driven and the old one closed by a *stoppting* which may be made of brick.

The result is that the area is divided up into a series of more or less rectangular blocks, the dividing lines being *bords* and *thirlings*.

To Ventilate a Chamber.—When gas is accumulating in the upper part of a chamber like opening, it may be removed by erecting a vertical brattice until it nearly reaches the roof. The air then sweeps along the upper part of the chamber and carries out the ac-

cumulating gas. In the lower portion of the brattice, a door is left for the passage of the miners.

Air Splitting.—From the formula on p. 181 we see that the quantity of air flowing along a level may be expressed $\sqrt{\frac{FA^2}{klc}}$ from which it follows that an air current would divide itself between two or more air ways inversely as the square root of their length or of their perimeters, other conditions remaining constant. Again with a given head or urging force, the quantity flowing will be greater by allowing the air to pass through several courses rather than by allowing it to pass through one course, the length of which is equal to the sum of the length of the branches. For this reason the air at the bottom of a downcast is *split* into several branches each of which circulates through its particular district and then returns to the upcast. Farther, by *splitting*, each set of workings receives air which has not been vitiated and in case of an explosion, a disturbance in one district is not likely to interfere with another district.

Brattices.—These are usually made out of $\frac{1}{2}$ or $\frac{3}{4}$ inch deals supported between temporary props.

A special kind of canvass is also very largely used for brattic-ing. The advantage of a brattice of this kind is that it can be very quickly constructed. Its liability to take fire is however a drawback to its use.

Stoppings.—When it is wished to completely isolate one portion of a mine from another, as for instance the *goaf* or old workings from places where workmen are actively employed, *dams* or partitions called *stoppings* are built in the levels. These are usually constructed of stone or brick and are made from 2 to 4 feet in thickness. To make them air tight they may be covered with tar or asphalt.

Where it is expected that a *stopping* might be required to resist great pressure, as for instance the effects of an explosion, it might be built of two curved walls, the intermediate space being filled with rubbish.

Doors.—In the roads along which traffic passes, the *stoppings* are replaced by movable doors. These are made to open in the direc-

tion of the loaded tub, and so hung that they swing to of their own accord. In order that the opening and shutting of these doors shall cause as little disarrangement as possible in the ventilating current, they are made in pairs, and placed at such a distance apart, that a train of tubs can pass through one of them and then close it, before opening the second.

In order to attend to these doors, more especially to see that they are properly closed, a boy called a "*trapper*" is stationed near them.

In some mines instead of using well fitting doors they use a square of canvass hanging like a curtain.

If this is tarred it may have an objectionable odour, and where fire damp occurs it is said that it might prevent the miners detecting the presence of such gas by its smell. In dry mines it forms a cheap and tolerably good substitute for wooden doors.

Self Acting Doors.—Instead of employing "*trappers*," self acting contrivances have been employed at several mines. One plan is to have two folding doors hung upon axes which are slightly inclined from the vertical. When a tub strikes these, they are forced open. After the tub has passed, they then swing together by their own weight.

By other contrivances the doors after being forced open, swing together by means of a falling weight.

As a rule these self acting doors have not hitherto been sufficiently successful to guarantee their employment.

Main Doors.—All main doors cutting off one portion of a mine from another are also made in pairs. If these become destroyed the ventilation of the whole mine may be stopped. So that they may not be tampered with they are usually kept locked.

Man Doors, are doors opening into the return air ways. They are always carefully locked, and can only be used by the overmen.

Dam Doors, are doors which are ready to be fixed in a level for the purpose of cutting off one portion of the mine from another, as for instance would be necessary in the case of spontaneous combustion.

Safety Doors.—At the time of an explosion it may happen that many of the doors in a mine are blown down and the ventilation at the moment when it is most needed becomes totally deranged. To obviate this, *safety or swing doors* have been devised. These are doors which swing upon a horizontal axis. Usually they are held up against the roof. When an explosion occurs the support which keeps them in this position is carried away, and they then fall by their own weight to stop up the level.

Some forms require to be loosened by hand. The plan seems to be good but Burat says he does not know a single case where they have been successful.

Crossings.—Wherever two currents of air have to cross each other it is well to allow the return current to pass above the ingoing one. This is best done by carrying one level over the other, and at such a height that sufficient material shall remain between them to resist the effects of an explosion which might tend to destroy the partition.

Sometimes one is carried over the other by brick or stone arching, or as at Wigan by means of arching made from boiler plate.

Conveyance of Air by Means of Pipes.—In small mines and for the purpose of ventilating single levels or openings which are not of any very great extent, pipes are very often used. These have usually a square cross section and are made of planks. At the joints they are covered with clay to make them air tight.

At one end, these pipes are made to taper so that they can be joined together by fitting one inside the other.

Pipes with circular sections are much better than those which are square and for this reason metal pipes are often used. The most common are those made from sheet zinc. These are circular in section and are joined together by placing the end of one inside the other, and then smearing the joint with a kind of putty.

Cast iron pipes have been used, but they are expensive Sheet iron pipes, are much better. Copper and lead have been also employed. Earthenware pipes are easily broken. Pipes made from *papier maché* have been tried, but these are destroyed by moisture. Canvass pipes strengthened by wooden rings may be employed to lead away a strong current.

In using zinc pipes all iron fastenings must be kept away, otherwise electric actions set in, and the zinc is rapidly destroyed.

Sometimes a notch or trench is cut along the side of a gallery, the face of which being covered with boards a pipe, known technically as a *ragglin* or *trumpeting* is formed. *Air sollars*, which have been spoken about on page 183 might be classed with pipes of this description.

ORE DRESSING.

Ore dressing is the separation of valuable minerals from the valueless materials with which they are associated when brought out of the mine. The usual processes are *reduction*, *sizing* and *sorting*. The ore is broken up to free the valuable mineral of a certain size, which is then sorted. The remaining ore is then reduced to a smaller size and more valuable mineral sorted out from the mixture,—*reduction*, *sizing* and *sorting* forming a cycle of operations. The word concentration is usually applied to certain processes by which the lighter and poorer portions of a mixture are gradually eliminated by washing.

The degree to which concentration may be carried depends upon local and other circumstances. For example.

1. The loss of valuable mineral is greater the higher the concentration. Valuable ores therefore should not be concentrated too closely.

2. The loss of metal which takes place during subsequent metallurgical processes, will vary with the degree and nature of the concentration.

3. The expenses connected with reduction processes which may be so much per ton or may vary with the degree of concentration.

4. Expenses connected with the processes of concentration itself.

5. The cost of transport before and after concentration.

At Mount Bischoff, Tasmania, tin ore is concentrated from 2 or 3 per cent up to about 75 per cent. To reduce the cost of transport by rail, the damp concentrates were dried. This resulted in a loss of the ore as dry dust, the value of which was greater than the extra cost of transporting the ore when damp.

Loss in Concentration.—The loss chiefly depends upon the fineness of pulverization, the degree of concentration attempted, the nature of the ore and its associated gangue.

Kustel gives as average losses, galena 15 to 20 per cent. Silver ores like sulphides 35 to 45 per cent. Gold 5 to 30 per cent. For the latter from 10 to 40 per cent might be a nearer approximation.

At Clausthal where the stamp ore averages 3 to 4 per cent argentiferous galena, the dead slimes often run 2 to 4 per cent of lead. Total loss in all dressing processes may be 12 to 16 per cent.

Metals and Minerals which require to be Separated.—

The following is abstracted from Gäetzschmann's Aufbereitung· Vol. 1 p. 17.

From Copper separate Iron ores (except spathic iron) lead.

„	Tin	„	Iron, copper and arsenical pyrites, blende, wolfram. Bismuth makes tin dull. Copper makes it brittle.
„	Lead	„	Arsenic makes it brittle, antimony hard.
„	Zinc	„	Lead.
„	Silver	„	Nickel, antimony, arsenical pyrites, carbonate of lead, galena if the silver is obtained by amalgamation.
„	Cobalt	„	Calc, manganese spar, hornstone, feruginous quartz, galena.

The methods of separation vary, thus a mixture of tin, iron, wolfram, molybdenum, blende, arsenical iron and copper pyrites, bismuth, antimony ores and earthy minerals may be reduced by washing to tin ore, wolfram, arsenical iron and copper pyrites. After roasting and reducing the sulphides to oxides, a second concentration reduces the mixture to tin ore and wolfram which are separated by a smelting process.

In many instances it is advantageous for two minerals to occur together, thus the smelting of magnetic iron is assisted by the presence of calcite, garnet, augite, hornblende &c. The smelting of lead, silver and copper ores is assisted by fluor spar. Spathic iron and heavy spar assist in smelting lead ores.

At Iserlohn in Westphalia, blende and iron pyrites are separated by roasting by which they are reduced to zinc oxide and ferric oxide. Subsequent jigging gives marketable products. At the Lintorf lead mines (Rhenish Prussia) blende and iron pyrites are separated by first

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passing through a disintegrator which breaks up the former more than the latter. The broken product in then sized.

Some ores are separated by means of electric magnets. Thus iron pyrites and spathic iron may by proper roasting be converted into magnetic oxide. A machine by Siemens 3½ feet long 2½ feet in diameter, treats 1 to 2 tons of ore per hour.

REDUCTION.

This section relates to the breaking or crushing of ore preparatory to concentration, to effect which satisfactorily the ore must be reduced to particles of a proper shape and size. If a ore is crushed too fine, during the subsequent washing processes the loss may be very great.

Rock Breakers.—Before fine crushing, the blocks of ore may be broken by sledges, or by a heavy single stamp weighing about 1500 lbs and giving 15 blows per minute or by some form of rock breaker.

Rock breakers of which Blake's is the best known type, consist of two heavy jaws ons of which is caused to approach and recede from the other. Sometimes both jaws are movable. If the width of a jaw at the top is 12 inches and the distance between the two jaws is 6 inches, the size of such a machine is described as 12 × 6.

The following table gives the sizes, the work that may be accomplished and the *H. P.* required for a series of Blake-Marsden rock breakers.

Size of machine at mouth	Approximate Product per hour.		Power required.	Total weight of machine.
	Road metal.	Fine Gravel.		
Inches	T. C.	T. C.	N.H.P.	T. C. Q.
10 × 8	4. 3	0. 20	3	4. 14. 1
12 × 8	5. 8	0. 25	3	4. 16. 0
15 × 8	6. 5	0. 29	5	5. 12. 1
15 × 10	7. 10	0. 32	6	6. 19. 1
20 × 10	10 0	0. 40	8	10. 5. 0
24 × 13	15 0	0. 50	10	14. 11. 3
24 × 17	16. 5	0. 55	14	14. 15. 0
24 × 19	17. 10	0. 60	16	18 0. 0

Fragments obtained from a machine having a mouth 10 inches × 2 inches, and moving about one eighth of an inch are suitable for jigging.

Quartz which has been reduced by a breaker may increase the efficiency of a stamp battery 20 to 25 per cent.

If a Rock Breaker produces dust likely to injure machinery in a mill it may be placed in a separate building.

There are many varieties of Rock Crushers in the market practically embodying the same principle as Blake's as the *Lancaster Rock Breaker* and the *Universal Crusher*. This latter crushes down to dust. In the *Meech crusher* the jaws are concave and convex.

The Comet Rock Crusher.—This consists of a conically shaped crusher head carried on a vertical shaft which below is ¼ to ¾ inch out of center. When this revolves the crusher head moves too and fro approaching and receding from the sides of the conical casing by which it is surrounded. The head does not turn while crushing the material which is fed between it and the outside lining. According to the size of the machine it is said that from 4 to 60 tons of rock may be crushed to the size of Macadam per hour.

Stamps.—When an ore has to be roasted after reduction, or when water is scarce, material may be stamped dry. Material by stamping is usually reduced to about 1½ to 4 mm. in diameter. In Australia experiments have shewn that for gold quartz, a preliminary roasting has sometimes resulted in a large yield of gold and in less wear to the machinery. The reports respecting the result of roasting are however often conflicting. Usually mine stuff is crushed with water so that the fine material passes through the gratings as pulp. We generally find five stamps working in one mortar.

Ten stamps in a mill usually constitute a battery.

The foundations of a battery consist of a series of heavy timbers laid horizontally in layers, the timbers in one layer being transverse to those above and below, the whole being bolted together by vertically placed bolts. Sometimes the mortars stand on the end of timbers placed vertically in the ground, 8 or 10 feet long and resting on heavy base timbers. Above the foundation and attached to it are the uprights held in position by struts and ties and horizontal cross pieces acting as guides.

Iron battery frames are used where there is a want of timber. In *Howland's battery* a number of stamps are placed in a circle.

In ordinary batteries the stamps which consist of stems with heavy heads are lifted by a cam, after which they fall and strike a blow. In certain patent batteries the stamps are lifted and caused to fall directly by steam as in a steam hammer.

In *Pneumatic stamps* an accelerated rate of fall is obtained by the expansion of air in a cylinder which had been compressed during the upwards motion of the stem. Steam stamps act somewhat similarly.

Mortars and Screens.—A cheap from of mortar may be made of heavy planking lined with iron plates, the bottom being composed of quartz fragments beaten down to form a hard bed. It is stated that such a bottom yields a fine material and the metallic materials are not beaten into scales which is a form disadvantageous for concentration. It is difficult to keep such mortars tight.

Ordinary mortars which are bolted to the mortar blocks are made of cast iron, 4 or 5 feet long and deep. They have circular dies beneath each head, placed 2 or 3 inches below the discharge. They are sometimes lined with iron plates which can be replaced when worn. At the upper side of the mortar there is a long narrow opening 3 or 4 inches wide for the admission of material to be crushed, and on the front side a screen or grate 12 to 18 inches deep through which the material is discharged.

With dry crushing the batteries are closed tightly with a box, and the dust is drawn from the mortars through a wire cloth grate or a slit, by a draught of air created by a fan or chimney and carried on to dust chambers where it is deposited. The dust may be assisted in settling and even drawn away from the batteries by a jet of steam. In wet crushing the ore is sometimes admitted at one end of the mortar and after passing underneath all the stamps which fall successively it is discharged at the other end. With this arrangement the stamps are of different weights and lifts.

The usual plan is to discharge on the long side of a mortar. Discharging on two sides or double discharge has been given up. Sometimes this is through a slit the size of which can be varied, through grates made of wrought iron bars, or through screens made of punched iron plates, brass or steel wire cloth. Sheet iron plates which are

made of the best Russian iron, are placed so that the side of the plate from which the holes were punched, is on the outside, as the holes are smoother on this side and slightly larger. This arrangement assists in preventing choking. Nevertheless it is necessary to repeatedly brush the screens.

By keeping the level of the water outside the screens equal to that in the mortar the tendency to choke is also reduced.

Screens are numbered from 1 to 10, number 1 being the coarsest. They are tacked on to a wooden frame, lined with strips of blanket.

Brass and wire cloth wears away rapidly.

Fraser and Chalmers manufacture annealed iron wire cloth from No. 1 mesh (No. 3 wire) to No. 80 mesh (No. 40 wire) and heavy steel tempered battery wire cloth No. 4 mesh (No. 12 wire) to No. 50 mesh (No. 34 wire).

Stems, heads and cams.—Usually the stems or lifters are made of iron circular in section but slightly coned at both ends. Length 10 to 20 feet and 2½ to 3 inches in diameter. Square wooden stems are used for cheapness.

The heads or sockets for round stems are cylindrical blocks of cast iron with conically formed holes at each end, one for the admission of the stem and the lower and larger one for the shoe.

The neck of the shoe is covered with strips of wood before being wedged into the head. This admits of its removal. Each end of the head is strengthened by wrought iron bands.

Shoes are made of white cast iron. They wear away at the rate of ½ to 1 lb. per ton of hard quartz.

Old forms of heads were rectangular in section and composed of one piece. Each stamp falls on a die of cast iron about 4 to 6 inches thick. The die is held in its place in a circular recess in the bottom of the mortar.

On each stem there is a lifter or tappet which in modern stamps is cylindrical in form. This is fixed on the smooth stem by a gib and keys. By the friction of the cams upon the tappets while lifting, the stem is caused to revolve. In wooden stamps where the tappets are simple wooden tongues there is no rotation, in consequence of which the effects of friction and wear are unevenly distributed.

The cam face is curved to form an involute of a circle which has a radius equal to the distances between the center of the cam and the center of the stem. Cams of this shape lift the tappet vertically and uniformly. Their faces are not oiled as that would reduce the friction too much, but instead, are covered with a composition of oil, tar, resin and tallow which is sufficiently lubricating and at the same time keeps the metals from actual contact.

Working Effect of Stamps.—(*Kustel*).

If p = the weight of a stamp in pounds.
h = its lift in feet,
f = the area of the stamp in square inches.
W = the working effect.
w = „ „ „ per square inch.

Then $W = p\,h$ and $w = \frac{p\,h}{f}$

For very hard rock $w = 6$ to 7 feet pounds.
„ hard rock $w = 5$ „ „
„ moderately hard rock $w = 4$ „ „

A stem 11 feet long 2¾ inches diameter weighs about 220 lbs.
Head 8 inches diameter 19 inches high „ „ 185
Shoe „ „ 90
Tappet „ „ 65
Total 560 lbs.

Stamps sometimes weigh 800 lbs. and each stamp requires about 1½ *H. P.*

If p = weight of a stamp in lbs.
h = lift in feet.
n = number of lifts per minute.
N = „ „ stamps in a battery.

Then the *H. P.* required $= \frac{N \times n \times h \times p}{33{,}000}$

On account of friction this quantity requires to be increased ⅓.

Arrangement, weight and lift of stamps.—Cams are arranged on a shaft so that the shaft experiences a uniform resistance. They are usually in groups of ten.

With a long line of cams on a single shaft, the repairs which may be required for one of them may necessitate the stopping of a series of batteries. For this reason separate shafts are used for each battery or each pair of batteries. Another principle to be observed in arranging cams is to make the order of lifts such that when one stamp falls, the adjoining stamps on the right and left are being lifted.

In two five stamp batteries with stamps 1, 2, 3, 4, 5, and 1′, 2′, 3′, 4′, 5′, the cams might be arranged to lift in the following order 1, 1′, 5, 5′, 3, 3′, 2, 2′, 4, 4′.

To hang up the stamps, that is to raise them above the range of the cams, while the cam is moving, a workman places on the face of a cam a piece of wood which causes the stamp to be lifted higher than usual, at which moment a hinged strut (a hanger) is placed beneath the tappet; the cam passes on and the stamp is left suspended on the hanger.

The lift and weight of stamp depends upon the quality of the ore. For fine crushing, high lifts, light stamps and deep mortars are used, while for coarse crushing, heavier stamps, low lifts and shallow mortars are employed.

Work done by stamps.—600 lb. stamps, with 75 blows per minute and No. 4 or 5 screens, will crush 1 to 3 tons per head in 24 hours.

In Cornwall about 1½ cwt of fuel are burnt per ton of stuff crushed.

Clayey ore requires more water than clean material,—10 lbs per minute for quartz. For quartz rocks each stamp requires about ½ to ¾ cubic foot of water per minute. One man can supply ore at the rate of 25 to 30 tons to batteries per 24 hours, providing that the material to be supplied is near at hand and of proper size. When the stamp requires ore, it may strike a lever, which in turn strikes the bottom of a hopper and a certain quantity of ore is caused to slide forwards. Such a feed is intermittent. A continuous feed may be obtained by causing a stamp to repeatedly lift a hopper or box which when it falls causes a small quantity of ore slide forward.

Amongst automatic feeders we have *Stanfords*, the *Tulloch* and *Challenge*.

With an automatic feed one man may attend to 80 to 150 stamps.

At Clausthal there are batteries with eleven heads falling in the following order, 6, 2, 10, 7, 11, 3, 9, 8, 4, 7, 5. The stems are 5 inches from center to center. The dies are half an inch broader than the shoes. Working 22 hours per day they last 10 to 12 weeks. The stamps weigh 440 lbs. and the drop is 8 inches.

There are 60 drops per minute. Each head yields 1⅓ tons of quartzose material $1^{mm.}$ in diameter : per day. Each battery requires 7½ cubic feet : of water per minute.

The objection to all forms of stamps is the large quantity ot pulp which is produced (90 per cent) and as this is rich in mineral, the loss in subsequent dressing operations is great.

When quicksilver is used in a battery to amalgamate gold, ⅛ to ½ oz of mercury is added on the feed side about every half hour.

Sometimes the long sides of the mortar are lined with amalgamated copper plates.

At the Mettacom mill, Austin, Nevada, it was found that with dry crushing, the gain in quantity crushed was more than proportionate to speed, as is indicated in the following table :

No. of Drops per minute.	Horse Power per stamp	Increase in Power.	Yield in tons per 24 hrs.	Increase of Yield.
50	1.36	.....	4½	.....
90	2.04	50 per cent	10	122 per cent
102	2.22	10 per cent	15½	115 per cent

Sholl's Recoil Stamper.—The stamp being worked by the movement of a spring beam, the blow has an elastic recoil. They reduce about 1 ton per hour.

Huntingdon's Oscillating Stamp Mill.—This consists of a bifurcated arm. On the bifurcations which are like an inverted Y are the shoes which rest on the dies. As the upper arm is oscillated the shoes alternately strike and slightly slide on the dies. It therefore crushes and grinds the ore.

Pneumatic and Elephant Stamps.—These stamps are said to crush 1 ton of quartz per hour. The cost is said to be 60 per cent less than Cornish stamps of the same power, and less fuel is consumed in their working. Repairs are however frequently required.

Sholl's Patent Direct Acting Pneumatic Stamps.—These stamps are manufactured in pairs, and are guaranteed to be of the capacity of 36 ordinary heads of cam and lifter stamps.

Kendall's Quartz Mill.—In this mill, the stamp stem is rotated. By its rotation the tappet, which is fixed upon it, rises up the cam, which is a plain curved incline. When it drops from this, a grinding action is produced.

McLeans Patent Alpine Crushing Machine.—Here 5 stamps work in a box, the from of which is the segment of a circle. Twelve cams or inclined plains are bolted on the rim of a horizontal wheel which, as it revolves lifts the stamps.

It is doubtful whether the last four mentioned forms of stamps have met with any marked success.

Price of Stamp Mills.—

The following prices are taken from a list issued by the Sandycroft Foundry.

No. of Stamps.	5 Stamp Mill	10 Stamp Mill.	15 Stamp Mill.	20 Stamp Mill.	30 Stamp Mill.
	£	£	£	£	£
Price of Mills.	180.	360.	540.	720.	1,080.
Price of Wood Framing.	75.	95.	170.	190.	285.
Price of Iron Framing.	110.	145.	190.	290.	435.
Price of Wood Foundation.	32 10s.	65.	97 10s.	130.	195.

Approximate Prices of complete Gold Mills, including Engines, Boilers, Stonebreaker, Stamps, Countergear, Copper Plates, Blankets, Amalgamating Pans, Settlers, Retort, &c.

5 Stamp Mill.	10 Stamp Mill.	15 Stamp Mill.	20 Stamp Mill.	30 Stamp Mill.
£	£	£	£	£
1,100.	1,800.	2,400.	3,100.	4,500.

Rolls.—Rolls for crushing ore are cylindrical in form, from 9 to 36 inches long and 18 to 24 inches in diameter. A pair of rolls are connected by gearing. For coarse crushing, the rolls are corrugated. The speed on the circumference is from 12 to 36 inches per second. They are employed for soft materials like limestone. The material after passing through one or two pairs of rolls is sifted in a trommel, and

the coarse material returned by a *raff wheel* to the upper side of the rolls for re-crushing.

Rolls are covered on the outside with a mantel or ring 1½ to 2 inches thick, of cast iron or steel, which, when worn can be replaced. The material treated is usually 1⅓ to 2⅔ inches in diameter. Tailings from jigging ⅝ inch in diameter are sometimes crushed. When material has been reduced to ¾ inch in diameter it may be jigged. Sometimes it is reduced to ⅓ inch diameter.

At Przibram, one pair of coarse rolls delivers stuff to two pairs of middle rolls, which supply one pair of fine rolls. The sizes obtained are 22, 14, and 9 mm. in diameter. The driving may be by belting.

The frame carrying a pair of rolls is massive and strongly bolted to a solidly built foundation. The bearings are arranged to slide in grooves and are kept in adjustment by springs, or by weighted levers. The result of this is, when any unusually hard material comes between the rollers, by compressing the springs or raising the levers, the rollers separate and the obstruction passes between them. So that the separation of the rolls may not affect the driving wheels these latter are connected with the axis of the rolls by a tumbling shaft. Involute teeth are used because they work truly at varying distances of their centers. For fine material it has been customary to reduce the diameter of the rolls, but this does not appear to be necessary. The feed may be automatic, as with stamps, the material by means of a stone breaker being first reduced to a uniform size, say not greater than 2 inches in diameter. Usually the ore is crushed wet.

If D = diameter of the rolls.
d = diameter of the material to be crushed.
s = the distance apart of the rolls or the diameter of the crushed material.

Then $$D > 18\,(d - s)$$

Resistance to rupture by compression per sq : centimeter.

Galena	45	Kilos.
Fluor spar	70	„
Iron pyrites	90	„
Blende	100	„
Quartz	100–300	„

DIMENSIONS AND PRODUCT OF CORNISH ROLLS (*André*)

Name of Mine.	Rolls.				Total pressure on Rolls.	Sifter or Tromel.				Diameter of Raff wheel.	Horse power.	Quantity crushed in 10 hrs.	Cost of crushing per ton.
	Diameter	Length.	Revolutions per min	Crushing area per minute.		Diameter.	Length.	No. of Hole.	Revolutions per minute.				
	ins.	ins.		sq. in.	cwts	ins.	ins.	sq. in.		feet		tons	pence
Grassigton Mines...	27	12	5½	5,593	91	21	48	6¼	37	14	...	80	...
Minera	14	14	8	4,920	73½	24	42	9	48	10	6	20	2½
Cwmystwith No. 1...	27	14	4	4,748	78	20	33	9	24	16	...	32	2½
Cwmystwith No. 2 ..	27	14	4½	5,341	85	24	36	9	24	16	...	35	2½
Goginan	30	14	5½	7,254	39	20	39	9	36	16	...	30	2¾
Cwm Erfin	27	14	7½	8,902	293	26	32	9	30	16	...	20	3
Lisburne No 1	27	15	6	7 632	180	22	36	12¼	30	16	...	42	2½
Lisburne No. 2	27	14	6	7,632	224	22	36	12¼	30	16	...	42	2½
Derwent	27	14	7	8,309	227	22	60	16	...	15	...	60	2¼
Goldscope	14	18	14	11,060	6	...	...	...	...	...	...	25	2½
East Darren.....	30	18	6	9,996	207	24	36	16	45	16	...	25	2¾
Cefu Cwm Brwyno.	20	13	5	4,080	84	20	48	16	27½	14	...	20	2½
Lisburne No. 3	18	16	8	6,432	169	22	36	25	30	16	...	42	2½
Llandudno	18	15	15	12,705	61	...	...	...	..	...	...	30	...
Wheal Friendship...	23	12	10	8,670	123	24	36	36	30	13	13	20	11½
Pontgibaud	25	12	12½	12,075	36	22	44	36	60	15	15	17	2½
Devon Great Consols	34	22	7	16,443	458	24	84	64	21	...	...	65	3¾

Grinding.—Ores are usually subjected to grinding processes when metals like gold or silver have to be separated from them by amalgamation, the amalgamation being sometimes conducted simultaneously with the grinding as it is sometimes conducted with the stamping.

The Arastra or Tehana.—The arastra is a circular excavation 12 or 16 feet in diameter lined with a paving of hard rock. In the center of this, there is a vertical axle carrying four or more horizontal arms to which by means of chains or ropes large grinding stones are fixed. As the axle and arms are turned the stones drag over the pavement, and reduce fine ore to a powder or pulp.

The Chili or Edge Mill. This is a machine which is commonly used for grinding cement. It consists of two rollers 3 to 6 feet in diameter standing on their edges in a horizontally placed iron basin. Either the edge rollers are caused to run round the inside of the basin over the material to be crushed, or the basin which may be pivotted at its center runs round beneath the rollers or runners.

Ball mills are a modification of the Chili mill, where balls are used in a basin instead of the rollers.

In some Pulverizers a large number of small balls are placed with finely divided ore in an iron cylinder which can be revolved. In *Thompson's Pulverizer* a single ball does the grinding.

The *Morey's Pulverizer* the balls work in a covered dish running on a vertical spindle.

In *Cylinder mills* the grinding is carried on between the side of a cylinder and a fixed surface.

Berdan Pans are largely used in Australia for gold amalgamation.

Horizontal stone or *iron mills* similar to the form used in grinding corn, are sometimes employed for dry grinding. They wear out quickly and are not very satisfactory.

Centrifugal Mills and Pulverizers.—In centrifugal mills material is supplied at the center of a horizontal disc provided with radial arms. As the material travels outwards, by the particles striking each other and the arms and finally the surrounding case, it is disintegrated. In *Carr's disintegrator* the material strikes a series of arms revolving in opposite directions.

In the *Sturtevant mill* two cylindrical heads are placed on the opposite sides of a hopper and are caused to revolve in opposite directions. It is simple and small.

A 4 inch Sturtevant is advertised to crush 100–400 lbs. of material per hour requiring 5 *H. P.* while an 8 inch mill crushes 1,000–2,900 lbs. and requires 13 *H. P.*

In *Jordan's Patent Pulverizers* two sets of four arms revolve in opposite directions close to each other.

Among other pulverizers we have the *Griffin* and *Lightning Pulverizers.*

In *Waring's Centrifugal Pulverizer* the work is done by 3 rollers which are pushed round the interior of a rotating cylinder. The *Frisbee Lucop mill* is another centrifugal roller pulverizer.

Among the newer machines used in the comminution of ore we have the *Dingey pulverizer*, the *Heberle mill*, the *Brink and Huebner disintegrator* and the *Schranz roller mill.*

The results of experiments made with three of these at Przibram representing the average of a year's work is as follows (*Kunhardt*).

Comparison for.	Order of Efficiency 1st, 2nd, 3rd, and 4th for the.			
	Heberle mill.	Dingey mill.	B. and H. Disintegrator.	Fine Rolls.
Minimum production of fine pulp.....	1	4	2	3
Labor, power, lubrication.	4	3	2	1
Wear per ton ore	1	3	4	2

In the Schranz roller mill, grinding action is avoided and but little fine pulp it produced. In reducing stuff of 5 and 1 mm. down to 2.4 mm. it gave as compared with a mill the following results expressed as percentages.

	3.2-2.4 mm.	2.4-1.6 mm.	1.6-0.9 mm.	1.9-.5 mm.	.5-.2 mm.	Less than .2 mm.	First pulp settlings.	Second pulp settlings.	Finest basin settlings.
Stamp battery.....	...	4.68	15.15	16.96	24.08	16.72	5.87	3.88	12.66
Schranz Mill	6.95	21.07	26,27	16.92	15.31	7.21	2.23	0.71	1 83

The Schranz Mill wears well, uses but little power and as seen from the above table yields good results.

Varney's quartz grinder consists of two conical grinders with bases downwards. The material is gradually reduced as it descends between the conical surfaces which at the top are separated for the introduction of the ore,

The Pneumatic Pulverizer.—In this machine two jets of superheated steam drive two streams of mineral particles into collission which pulverize each other by their impact.

Pan Amalgamators.—In Pan amalgamators, amalgamation and grinding may be carried on simultaneously. This however is not usual. The charge is wet, definite in quantity and is worked upon for a specified period. According to the form of the grinding surface Kustel divides pans into four classes, 1st plane circular, 2nd conical, 3rd tractory conoidal, 4th vertical mullers.

An ordinary amalgamator consists of a circular iron pan rising from the center of which there is a hollow cone. The bottom of the pan is lined with a disc or dies of white cast iron. The sides may be of wood. Passing up the hollow cone there is a shaft which projecting at the top, carries by means of arms or a hollow cone enveloping the pan cone, the revolving grinding surface or muller which is faced with shoes.

Usually the muller can be raised from the bottom of the pan by screwing it up the central shaft, or the shaft itself which is driven by gearing beneath the pan, can be lifted by a lever. In this way the grinding power of the muller can be regulated.

About 600 to 800 lbs. of stuff are treated at a time. Coarse sand which is mixed with water to form a pulp, takes to grind and amalgamate about 4 hours, finer material 2 hours. About 5 *H. P.* is required per pan. By rapid motion of the mullers (65 to 85 or even 100 revolutions per minute) the pulp is thrown outwards. At the periphery of the pan it comes in contact with properly placed curved plates, which guide it back towards the center of the pan.

To assist amalgamation the pulp may be kept hot by a jet of steam or by a steam jacket on the bottom of the pan. In *Rae's patent electric system* of amalgamation, a current of electricity is passed through the pulp, with the result, it is said, of preventing the fouling and consequent loss of quicksilver. Several electrical arrangements which are said to improve and accelerate amalgamation, have been used in connection with stamp work and pan amalgamation. Each pan requires 10 to 100 lbs. quicksilver. One hundred pounds of mercury dissolve ¾ to 1½ oz. of gold. Among pans with plane circular grinding surfaces, we hvae the *Knox*, the *Varney* and the *Wheeler* pans.

The pans with conical mullers may have the bottom of the pan inclining downwards towards the center as in the *Hepburn* and *Petersons* pan, or as in the *Belden* pan inclining upwards towards the center.

In the former pan the pulp tends to flow inwards without the assistance of guide plates.

In the *Excelsior* and other pans the form of the base of the pan and mullers is conoidal.

The *Combination Pan* may be taken as a standard form.

In *Hinkle and Capp amalgamator and grinder*, perpendicular mullers are pressed laterally by centrifugal force against the perpendicular sides of the pan.

From grinding pans the pulp passes to *settlers* where it is thinned by the addition of water, and the quicksilver and amalgam, which has become finely divided, is allowed to settle.

A ten stamp silver mill requires about six pans and three settlers.

A pan holds from one to two tons of pulp. A 5 foot pan weighs 6,500 lbs.

The Bennett Amalgamator.—This is used for placer works. The ground is scooped up by a dredger and passed into a trommel

revolving partly under water. Here the stones are separated from pay dirt which falls into a water tank where jets of water cause it to pass over amalgamated plates. It requires 150,000 gals. of water per 1,000 cubic feet of earth. It is said to work placer ground for 1 to 3 cents per cubic yard, the usual price being 3 to 9 cents.

CONCENTRATION.

The simplest method of concentration is washing by hand in a pan. This method is carried out mechanically in several concentrators where the hand movement is closely imitated by machinery as in *Kew and Jones* concentrator used for the waste from Berdan pans and cleaning pyrites.

In ordinary concentration the processes which are followed after reduction are generally as follows.

1. For coarse stuff. (*a*) a classification into equally sized grains by sieves. (*b*) sorting into equally falling grains by machines like jigs.

2. For fine stuff. (*a*) a sorting into equally falling grains by apparatus like pointed boxes. (*b*) a classification by the action of a thin stream of water carrying the material down an inclined surface.

Varieties of Materials.—The different varieties of material yielded during different processes depend partly upon the nature of the material and partly upon the method of reduction.

The following table shews the percentages of different materials obtained by various operations.

	Smalls from mine.	Rolls.	Stamps.
	p.c.	p.c.	p.c.
Nut (Stufen).....	74	—	—
Pea (Graupen)	15	70	32
Sand (Gries)	7	20	32
Meal (Mehl).....	4	10	32
Pulp (Staub).....	4	10	36

From roils we may obtain.

Round grains 50 per cent.
Flat „ 25 „ „
Long „ 25 „ „

These percentages however depend upon the cleavage of the minerals, thus.

	Round grains.	Flat grains.	Long grains.
	p.c.	p.c.	p.c.
Galena.....	60	25	15
Quartz.....	40	30	30

SEPARATION AND SIZING.

Separation of Gangue from Ore.—The ore is sometimes roughly broken at the mine, for which purpose large steel sledges 18 to 25 lbs weight are employed. It is then roughly picked, the result being *attle*, *rich ore*, *middle ores* and *smalls*.

Outside the mine there are the processes of *ragging*, *spalling* and *cobbing*, each of which consists in breaking the rock and picking. Ragging hammers may be 6 or 8 lbs. weight, while those for spalling are two pounds weight. The extent to which these processes are carried depends on the nature of the ore and the price of labour. With high wages it might be more economical to send an ore at once to the mill, while in countries where labour is cheap it might be more economical to first pick out the more worthless portions, and to separate certain ores and minerals from each other which otherwise would be detrimental to one another during concentrating and metallurgical processes.

Cobbing hammers are 1½ to 2 lbs weight and have one end sharp edged. The ore is broken on dies which, if the ore has a tendency to fly may be surrounded by iron rings The smalls are usually jigged.

Sometime ores may require to be specially picked, and specially arranged tables at which the pickers sit are required. Before the ore comes to the pickers it is washed. Picking tables are sometimes made to revolve so that the minerals to be sorted pass before several pickers.

Some ores, like nickel ores, can only be recognized as valuable after having been exposed to the weather. Certain iron ores may be exposed to the weather so that they may be readily separated from adhering gangue. The result of these operations is to give worthless material, first class ore or *prills* ready for reduction, middle class ore or *halvings* requiring to be crushed and concentrated, and a third class ore or *smalls* ready to be jigged.

In picking ores, crushing ores are usually separated from cobbing ores. The former may be subdivided into *roll rock* and *stamp rock*. All of them may be subdivided according to their mineral constituents. Thus it would be well to separate galena (SG 7.5) from cassiterite (SG 6.8) or blende from copper pyrites, and these again from barren rock.

Washing and Sizing.—By raking the ore to and fro upon a grate upon which water is falling, the smalls pass through the grate and the remainder which is roughly washed is prepared ready to be picked. Several gratings with different sized holes may be employed in conjunction.

The ore coming from the mine may be tipped on grates having opening, about 2¼ inches wide, the lump ore passing over the grate and the finer stuff through it. The percentage of fine ore depends partly on the nature of the ore and on the method of working. Where explosives are employed the percentage of fine ore may be 50 or 60 per cent of all the ore. (*Kunhardt*).

Material 1¼ to ⅛ inch diamete is usually separated into several sizes for jigging. In dressing copper pyrites in Cornwall such material however is only divided into two classes.

In *sluices* which are inclined troughs about 2 feet wide and 4 to 10 feel long, ore is washed and concentrated.

The *Laüterrinne* is a trough 2 feet broad and 12 feet long, inclined 2 inches per foot. Material and clear water is supplied at the upper end, and as the material is raked downwards it is washed. At the lower end it passes over a series of sieves arranged in a step like series.

Riddles.—Riddles of various sizes and shapes may be used for washing and sizing. Oblong square riddles suspended by four chains or rods, two of which by means of screws may be altered in length, can be swung backwards and forwards by hand, or by cams, and caused to strike a block. When ore is placed on such a riddle a portion of it

passes through the grating while another portion travels forward to be discharged at the end.

If water is supplied on the stuff, the ore is washed. Several of these riddles of different sizes are employed together. Jarring or swinging riddles may be inclined and have a vertical motion.

The latter however are not so effective as the former, the motion tending to choke the holes. The riddles ought to be arranged so that the stuff first falls on the coarsest riddle. If it falls on the finest riddle first, the gratings will be rapidly destroyed. The less the inclination the slower the passage of the ore and the more effective the sifting. Different sized riddles ought to have different inclinations.

Size of holes $\frac{1}{15}$ to 1 inch in diameter. Length of sieve 3 to 7 feet. Inclination 1 to 5 inches per foot. Speed 30 to 70 strokes per minute. Water required, 3 to 16 cubic feet per minute. Sieves last 2 to 12 weeks (*Kustel*). Three to four *H.P.* work the heaviest riddles. Shaking riddles with 150 to 200 blows per minute give a cleaner and better separation than trommels where the effective screening surfaces is small and the shocks due to the fall of the material in the trommel are feeble and not more than 60 per minute. Riddles however require more power and wear more quickly than trommels.

Cradles.—The cradle or rocker is as lightly inclined coal scuttle shaped box resting on rockers. The stuff, for example auriferous drift, is supplied on a tray or sieve at the upper end of the box, and while the box is rocked by hand, water is poured on the stuff, the finer parts of which are carried inside and flow down the inclined bottom. Crossing the bottom are strips of wood, behind which particles of gold are arrested. It is a combination of a sieve and a moving sluice.

Trommels.—Trommels may be described as cylindrical, prismatic or conically shaped sieves which are used for sizing and washing. For stuff to travel through a cylindrical trommel the central shaft on which it is carried must be inclined, or if horizontal, the inside of the trommel must be fitted with a spiral. The common form is the conical trommel, along the inside of which the materials travel even when the shaft is horizontal. Conical trommels require less motive power and are easier to manage than those which are cylindrical. For coarse work trommels are constructed of iron bars built together at spaced distances to form a drum, or of perforated boiler plate or cast iron. If the material which has to pass through the trommel is of a clayey nature the inside

of the trommel is fitted with a spiral arrangement of spikes. For washing purposes water is supplied inside the trommel either from a hollow axle or separate pipe, or the trommel revolves partially in a tank of water.

At the upper end of a trommel there is a conical collar to prevent stuff from falling out, and at the lower end another collar which widens outwards to assist the discharge.

For sizing, the trommels are made of sheet iron, copper plates, copper or brass wire cloth.

The slope inside a trommel will be less, the more washing a material requires. One in twelve is usually a sufficient fall. Length 9 to 12 feet, and diameter 4 to 5 feet. About 10 to 20 revolutions per minute. Inclination of axis 3° to 6°.

Trommels of different grades like sieves, may be arranged in series following each other longitudinally, or be placed concentrically, and grains of different sizes obtained.

Jets of water playing upon the outside of a sizing trommel prevent the choking of the holes. Clayey and slimey materials ought not to be admitted.

One separating and four or five sizing trommels, size about 25 tons of smalls in twelve hours (*Kustel*).

To obtain the best wearing of the sieves, the stuff ought to pass through the coarsest first. Sometimes stuff may be separated into four groups by four sizes of decreasing holes, and subsequently each group may be divided into four classes by a system of sieves with increasing holes.

Patent trommels with concentrically arranged sieves are difficult to repair and usually heavy.

Trommels of punched sheet metal usually have round holes but those with square holes do better work.

At Clausthal it has been found that copper sheet trommels which do not rust, last twice as long as iron trommels They cost 80 per cent more than the iron trommels. For coarse screens where rusting is of little consequence iron or steel is used. The ore may be fed from a pointed hopper from which the ore falls upon a rectangular oscillating pan which throws it forward to the trommel. A trommel 3½ feet in diameter and 9 to 12 feet long, sizes 7½ to 8 tons of ore per hour.

Trommels may be set in steps side by side, one drum in front of another, or on a single inclined axis. In the latter case they may be conical in form and so coned that they have a slight dip at their lower end.

Dimensions of Sieve Holes.

The following table is taken from Kunhardt's "Ore Dressing in Europe." It is similar to Rittinger's table.

Diameter of grains.		Wire cloth.		Name.	
Millimetres.	Inches.	Mesh.	No. of Wire.		
.64	2.56 = $2\frac{1}{2}$	3 in.	00	Large	No. 1 Nut (*Stufen*)
45.2	1.80 = $1\frac{3}{4}$	$2\frac{1}{8}$	0		
32	1.28 = $1\frac{1}{4}$	$1\frac{1}{2}$	2	Small	
22.6	.90 = $\frac{7}{8}$	$1\frac{1}{8}$	4		
16	.64 = $\frac{5}{8}$	$\frac{13}{16}$	7	Large	No. 2 Pea (*Graupen*)
11.3	.45 = $\frac{7}{16}$	$\frac{5}{8}$	8		
8	.32 = $\frac{5}{16}$	No. $2\frac{1}{2}$	12	Small	
5.6	.22 = $\frac{1}{4}$	„ $3\frac{1}{2}$	14		
4	.16 = $\frac{3}{16}$	„ $4\frac{1}{2}$	15	Coarse	No. 3 Sand (*Gries*)
2.8	.11 = $\frac{1}{8}$	„ 6	16		
2	.08 = $\frac{3}{32}$	„ 8	18	Fine	
1.4	.055 = $\frac{1}{16}$	„ 12	22		
1	.04 = $\frac{1}{25}$	„ 16	25	Coarse	No. 4 Meal (*Mehl*)
.71	.028 = $\frac{1}{32}$	„ 24	28		
.5	.02 = $\frac{1}{50}$	„ 35	31	Fine	
.35	.014 = $\frac{1}{64}$	„ 50	38		
.25	.01 = $\frac{1}{100}$	„ 70	38	Pulp (*Staub*)	

It will be observed that alternate numbers of the above series progress in geometrical ratio, or taking them successively one is 1 411 times the other. In some mills the factor of the geometrical ratio is 1.35 in others 2. The volumes of the grains which pass successive screens are 2.828 times each other.

The reason for proportioning the holes in sieves in this manner will be understood from the section treating of the relation between the diameters, volumes, weights and densities of equally falling particles.

If d is the diameter of a hole in a sieve, then the distance between the centers of the holes $= \frac{3}{2} d$, and for fine sieves $2 d$.

For fine material, sieves are made out of copper or brass wire, the holes being square.

Nature of Particles Required for Concentration.—Ore particles subjected to processes of concentration are usually either those which are of *equal sizes* or those which fall though a column of water with *equal velocity*.

The operation of jigging is dependent upon the different rates at which particles of the same size but of different specific gravities fall through a column of water.

All particles treated in a given jig ought therefore to be of the same size.

The lighter particles of *equally falling* stuff are larger than the denser particles, and therefore if placed upon an inclined table over which a film of water is flowing are more easily moved than the small denser particles which only receive an impluse from the film of water immediately in contant with the surface of the inclined place which more or less adheres to its surface.

Separation of reduced material into equally falling particles may be accomplished by one of the following methods.

1. By allowing the material to fall in still water.

2. By allowing the material to fall in a stream of water rising vertically.

3. By combining methods 1 and 2.

4. By the action of a horizontal stream of water.

Fine sands are sometimes treated on fine jigs. Meal is usually reated on percussion tables and pulp on round tables.

THEORY OF CONCENTRATION.

(a) The Movement of Single Particles.

1. Carrying power of current.

Let d be the diameter of a body lying on the bed of a stream with a velocity V, then the pressure exerted by the water tending to move the particle is:—

$$a\ V^2\ d^2$$

where a is a constant.

The pressure of the body on the bed of the stream is proportional to

$$d^3\ (\delta - 1)$$

where δ is the density of the body and 1 the specific gravity of water.

The frictional resistance of the body to be moved, where b is a constant, is therefore

$$b\ d^3\ (\delta - 1)$$

Equating this with the moving force, then

$$a\ V^2\ d^2 = b\ d^3\ (\delta - 1)$$

and collecting the constants under the symbol C.

$$V^2 = c\ d\ (\delta - 1)$$

or the maximum *diameters* of bodies of equal densities that can be carried along are proportional to the square of the velocity of the current.

Cubing both sides and changing the symbol of the constant to k

$$V^6 = k\ d^3\ (\delta - 1)^3$$

which shows that the maximum *weights* (which are proportional to d^3) of different bodies of equal densities that can be carried along, are proportional to the sixth power of the velocity of the current.

The application of this law to the regulation of the flow of water in sluices is self evident. It must be remembered that it is usually the bottom velocity of a stream that has to be considered, and farther that ordinary rocks lose from a half to one third of their weight in air when immersed in water.

As an illustration of the above law the following is quoted from the observations of David Stevenson.

A velocity of 3 inches per second just moves fine clay.

,,	,,	6	,,	,,	,,	lifts find sand.
,,	,,	8	,,	,,	,,	lifts coarse sand.
,,	,,	12	,,	,,	,,	sweeps along fine gravel.
,,	,,	24	,,	,,	,,	rolls along rounded pebbles one inch diameter.
,,	,,	36	,,	,,	,,	sweeps along slippery angular stones the size of an egg.

Von Sparre says that the diameters of particles of the same specific gravity deposited in shallow currents are directly proportional to the fourth roots of those velocities.

In considering questions relating to the movement of particles in a fluid like water, Pernolet and subsequent investigators like Sparre and Rittinger, assume the particles to be sufficiently far separated that the movement of *any one* is unaffected by the proximity of its neighbours and it is on assumptions of this nature that the following formulæ taken from *Rittinger's Aufbereitungskunde* are based. The practical application of these formulæ in the Continental System of dressing is extensive, and for this reason, and because they may be regarded as introductory to formulæ which follow, where the movement of particles *ên masse* is considered, they are here given. As these formulæ are but abbreviated extracts from Rittinger's work, students are referred to the original text.

2. The resistance of fluids to bodies falling through them.—

Pressure per square meter exerted by a stream of water rising vertically with a velocity of 1 meter per second.

For a horizontal plain.

$$a = 51 \text{ Kilos.}$$

Against the horizontal head of a wedge:

$$a_1 = \frac{a}{4\frac{h^2}{b^2} + 1}$$ where. b = the breadth of the head. h = the height of the wedge.

Against the horizontal head of an equally sided wedge,

$$a_1 = \frac{a}{4}$$

Against the horizontal base of a right angled wedge.

$$a_1 = \frac{a}{2}$$

Against the horizontal base of a cone.

$a^2 = \frac{ar^2}{r^2 + h^2}$, where r is the radius of the base and h the height of the cone

Against the horizontal base of a right angled cone.

$$a_2 = \frac{a}{2}$$

Against the horizontal base of a hemisphere

$$a_3 = \frac{a}{2} = 25.5 \text{ Kilogr.}$$

In the following formulæ.

d = the diameter of the particle or sphere in meters.
δ = the density of the particle.
Δ = the density of the fluid, (for water $\Delta = 1$).
$g = 9^{m}.809$ acceleration due to gravity.
$\gamma = 1000$ kilogr, the weight of a cubic meter of water.
t = the time in seconds during which a particle falls.
v = the velocity of the body in meters per second.
s = the distance traversed.

3. The fall of a small particle through a fluid at rets is approximately.—

$$v = C_1 \sqrt{\frac{d(\delta - \Delta)}{\Delta}} \qquad \left(C_1 = \sqrt{\frac{2\gamma}{3a_3}} \right)$$

For small spherical bodies $C = 5.11$.

For small flat grains it may $= 1.92$.

A mean value for irregular grains $= 2.44$.

For a small spherical particle falling through water after it has reached uniform motion, its velocity is approximately.

$$v = 5.11 \sqrt{d(\delta - 1)}$$

The following table gives the velocity in meters per second attained by grains of galena and quartz falling through a column of water. The size of the grains follows the Rittinger's sieve scale, the velocities are calculated from the formulæ.

$$v = 2.44 \sqrt{d(\delta - 1)}$$

Size d in meters.	Velocity of fall in meters.	
	Galena $\delta = 7.5$	Quartz $\delta = 2.1$
.064	1.571 } I	.780 } II
.0452	1.322 }	.671 }
.032	1.112 }	.551 }
.0226	.922 } II	.457 } III
.016	.786 }	.390 }
.0113	.661 }	.328 }
.008	.556 }	.276 }
.0056	.463 } III	.230 } IV
.004	.392 }	.195 }
.0028	.327 }	.161 }
.002	.278 }	.137 }
.0014	.232 } IV	.115 } V
.001	.195 }	.097 }
.00071	.165 }	.082 }
.0005	.138 }	.073 }
.00035	.116 } V	.058 } VI
.00025	.098 }	.048 }
.000125	.069 }	.034 }

4. The space a small particle describes in an extremely short time, (say .01 to 1 second) in still water is approximately given by the formula.—

$$s = \left(1 - \frac{1}{\delta}\right)\frac{gt^2}{2}$$

It is therefore only dependent upon the density of the particle. Two equally falling grains might therefore be separated if the height

of fall is extremely small, and jigs may be used as hydraulic sizing apparatus.

Von Sparre gives the period of acceleration for particles 16 mm. in diameter as 1 second, and if 1 mm. in diameter as .25 second.

The following table gives the space described by small particles falling through still water during extremely short intervals of time.

Time Falling in Seconds.

d Mill.	δ	.01	.02	.03	.05	.10	.15	.20	.25	.30	.50	1.00
	Space described in Meters.											
4	2.6	.0003	.0012	.0026	.0069	.0229	.0425	.0629	.0831	.1035	.1842	.3898
4	7.5	.0006	.0019	.0038	.0102	.0365	.0711	.1107	.1489	.1890	.3556	.7532
.25	7.5	.0004	.0016	.0036	.0091	.0272	.0475	.0678	.0881	.1084	.1896	.3927

5. Relation between the diameters, volumes and weights, and the densities of equally falling particles.—

Spherical grains of different diameters and densities which fall with equal velocities in water or other fluids are called equally falling grains.

Then if
d_1 and d_2 = the diameters,
δ_1 and δ_2 = the densities,
V_1 and V_2 = the volumes,
P_1 and P_2 = the weights,

of two spherical particles. Then if these fall through a given column of water in equal times.

Then by (2)
$$\frac{d_1}{d_2} = \frac{(\delta_2 - 1)}{(\delta_1 - 1)}$$
$$\frac{V_1}{V_2} = \frac{(\delta_2 - 1)^3}{(\delta_1 - 1)^3}$$
$$\frac{P_1}{P_2} = \frac{\delta_1 (\delta_2 - 1)^3}{\delta_2 (\delta_1 - 1)^3}$$

For example for equally falling grains of quartz and galena:

$$d_1 = 4\ d_2$$
$$V_1 = 64\ V_2$$
$$P_1 = 22\ P_2$$

If Δ = the density of the fluid, the first of the above equations becomes.

$$\frac{d_1}{d_2} = \frac{(\delta_2 - \Delta)}{(\delta_1 - \Delta)}$$

Calling d_2 the diameter of a spherical grain of galena with density $\delta_2 = 7.5$, and d_1 the diameter of a similar grain of quartz with density 2.6, for these to fall equally.

In air ($\Delta = 0.00125$)	$d_1 = 2.88\ d_2$
In water ($\Delta = 1$)	$d_1 = 4\ d_2$
In zinc sulphate ($\Delta = 1.5$)	$d_1 = 5.45\ d_2$

To separate galena and quartz in machines when the separation depends upon the velocity with which different grains of these materials fall through water, the broken material are according to the continental system of jigging (but not according to the English) classified by sieves into sizes, the maximum size of the grains in each division being in the ratio, 1, 4, 16, 64, &c.

6. The suspension of a body in a stream of water rising vertically.—

The velocity of the stream which holds a body in suspension is equal to the velocity which the body would acquire in falling were the water at rest.

For a spherical body as shewn in (3):

$$V = 5.11\ \sqrt{d\,(\delta - 1)}$$

The following table gives the velocity in meters per second of the upward current which holds different round bodies of different diameters in suspension.

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Material.	Density.	Diameter in Millimeters.							
		10	8	6	4	3	2	1	½
		Velocity in Meters per second.							
Gold (with Silver).	15.0	1.93	1.72	1.50	1.22	1.05	.86	.61	.32
Galena	7.5	1.31	1.17	1.02	.83	.72	.59	.41	.29
Iron Pyrites.....	5.0	1.03	.92	.79	.65	.56	.46	.32	.23
Quartz	2.6	.63	.56	.49	.40	.35	.24	.20	.14
Coal	1.3	.28	.25	.22	.17	.15	.12	.08	.06

With the formula $V = 2.44\sqrt{d(\delta - 1)}$ the following table giving the velocity necessary for the suspension of irregularly formed grains has been calculated.

Substance.	Density.	Diameter of seive in Millimeters.							
		10	8	6	4	3	2	1	½
		Velocity in Millimeters.							
Gold (with Silver).	15.0	.90	.80	.70	.57	.50	.40	.29	.20
Galena	7.5	.62	.55	.48	.39	.34	.28	.20	.14
Iron Pyrites.....	5.0	.49	.43	.38	.30	.26	.22	.15	.10
Quartz	2.6	.30	.27	.24	.19	.17	.14	.10	.07
Coal	1.3	.13	.12	.10	.08	.07	.06	.04	.03

7. Movement of a particle in a rising stream of water.—

The velocity v which a body will attain in a rising current, is approximately the velocity of the current C, minus the velocity necessary to hold the body in suspension or,

$$v = C - c.$$

8. The space a particle describes in an upward current of water in an extremely small interval of time (say .01 to 1 second) is approximately given by the following formula.—

$$s = \left(\frac{\varphi C^2}{d\delta} + \frac{1}{\delta} - 1\right)\frac{gt^2}{2}$$

where $\varphi = \frac{3a_3}{2\gamma}$

From this it follows that the smaller of two equally dense grains rises the higher, and for two equally sized grains the least dense rises the higher. The denser falls the quicker. If the bodies do not rise the denser will sink lower than the less dense as in quiet water.

Of two equally falling grains, at the commencement the smaller and denser moves higher than the greater and less dense. The upward stream therefore tries to place equally falling particles in the reverse order to what they would have in ordinary sieve work providing that these particles are free to move. If however the velocity of the descending current is small and equally falling particles are allowed to fall, the denser particle would fall quicker than the less dense as in still water.

9. Movement of a particle during an extremely small interval of time in a descending stream of water.—

Before a body attains the velocity of a descending stream C, it has an acceleration.

$$G = \frac{B}{A}[1 + A^2 (C - v)^2]$$

During this period it has a velocity,

$$V = C - \frac{1}{A} \tan (\tan^{-1} AC - Bt)$$

And when $C = V$.

$$t = \frac{\tan^{-1} AC}{B}$$

The distance described until the commencement of this is,

$$s = \frac{1}{AB}[AC \tan^{-1} AC - \tfrac{1}{2} \log n (1 + A^2C^2)]$$

The time taken for particles of quartz and galena to attain the velocity of a descending stream and the space they describe in attaining that velocity are given in the following two tables.

Velocity of stream in Meters.	Quartz SG 2.6 Diameter in mm.			Galena SG 7.5 Diameter in mm.		
	1	4	16	1	4	16
	Times in Seconds.					
.25	.030	.037	.040	.026	.029	.029
.50	.040	.060	.075	.043	.053	.057
.75	.044	.073	.101	.052	.072	.083
1.00	.047	.081	.121	.057	.085	.106
	Distance in Meters.					
.25	.0043	.0049	.0051	.0034	.0036	.0037
.50	.0133	.0173	.0196	.0123	.0139	.0145
.75	.0239	.0340	.0416	.0242	.0295	.3206
1.00	.0352	.0532	.0693	.0379	.0493	.0054

In the above expressions,

$$A = \sqrt{\frac{3 a_3 \Delta}{2 \gamma d (\delta - \Delta)}}$$

$$B = g \sqrt{\frac{3 a_3}{2 \gamma}} \cdot \frac{\sqrt{(\delta - \Delta)\Delta}}{\delta \sqrt{d}}$$

$a_3 = 25.5$ kilogr,

A formula which *approximately* expresses the distance described by a particle falling in a downward current during a short interval of time may be written :—

$$s = \left(1 + \frac{3 a^3 C^2 - 2 \gamma d}{2 \gamma d \delta}\right) \frac{g t^2}{2}$$

For two *equally sized grains* the denser sometimes goes over a greater distance sometimes over a less, thus if

$$C < \sqrt{\frac{3 g \delta}{3 a_3}}$$

the denser goes first. In the opposite case the denser remains behind.

In the case

$$C = \sqrt{\frac{3\,g\,\delta}{3\,a_3}}$$

the two equally sized grains fall equally without any regard to their difference in density.

For two equally dense particles at first the smaller falls the quicker and traverses a greater distance.

For two equally falling grains, the denser goes first and indeed even much quicker than in still water, from which it follows that a downward current may be extremely useful in separating equally falling particles.

10. The fall of a particle in a stream moving horizontally with a velocity C.—

Let x = the distance moved horizontally in the time t.

y = the distance moved vertically downwards in the time t.

Then approximately for spherical particles,

$$x = C\,t$$

$$y = C_1\,t\,\sqrt{\frac{d\,(\delta - \Delta)}{\Delta}}$$

when C_1 and Δ have value as before.

For irregular particles where D is the size of sieve.

$$x = \frac{y\,C}{C_3\,\sqrt{D\,(\delta - 1)}}$$

Here the body follows a straight line diagonally downwards.

For two bodies of different size and density we have

$$\frac{x_1}{x_2} = \sqrt{\frac{D^2\,(\delta_2 - 1)}{D_1\,(\delta_1 - 1)}}$$

from which it follows, that for equally falling bodies where

$$D_2 (\delta_2 - 1) = D_1 (\delta_1 - 1),$$

therefore also $x_1 = x_2$

and these equally falling bodies reach the bottom at the same place. A horizontal stream of water may therefore be used to obtain equally falling grains.

11. Particles in flowing water on an inclined plane.—Let

φ = Inclination of the plane.
s = Coefficient for sliding friction.
b = Breadth of base of a prismatic body, in meters.
d = Height of base in meters.
k = weight of particle in kilos.
r = The resistance of sliding friction.
w = „ „ „ rolling „
c = „ velocity of the water in meters per second.
f = The section of the nearly spherical particle at right angles to the current.
a_4 = Experimental value for pressure of water with a velocity of 1 meter on a surface 1 square m. in area (see p. 241).

Then

$$r = s \frac{k}{\delta} (\delta - 1) \cos \varphi$$

$$w = \frac{b}{d} \cdot \frac{k}{\delta} (\delta - 1) \cos \varphi$$

According as r is $>$ or $<$ w so will the body slide or roll.

The velocity to balance sliding fiction is:

$$c = \sqrt{\frac{k(\delta - 1)}{a_4 \delta f} (s \cos \varphi - \sin \varphi)}$$

For polygonal bodies nearly spherical:

$$c = \sqrt{\frac{2(\delta - 1) d \gamma}{3 a_4} (s \cos \varphi - \sin \varphi)}$$

The velocity to balance rolling is:

$$c = \sqrt{\frac{k(\delta - 1)}{a_1 \delta f}\left(\frac{b}{d}\cos\varphi - \sin\varphi\right)}$$

And for polygonal bodies:

$$c = \sqrt{\frac{2(\delta - 1)d\gamma}{3a_4}(\mu\cos\varphi - \sin\varphi)}$$

where $\mu = \frac{b}{d}$

From the above we see that when two different particles are carried along (sliding or rolling) the relationship of velocities of current is as follows:—

$$\frac{c_1}{c_2} = \sqrt{\frac{(\delta_1 - 1)d_1}{(\delta_2 - 1)d_2}}$$

From which it follows that equally falling bodies can not be separated by a stream of water either according to there density or according to their size. A flowing stream of water can however be used to classify or separate according to density in the case of equally sized bodies, or it can be used for separation according to size in the case of approximately equal density in the particles.

Equally falling grains may be separated into *coarse* and *fine* in a stream with decreasing velocity, or according to *density* in a very thin stream (see p. 262).

12. Influence of centrifugal force upon particles falling in water.—

Let

k = the weight of particle.
v = the velocity of rotation.
r = the distance of the particle from the axis of rotation.
g = force of gravity.

$$\text{Centrifugal force } F = \frac{kv^2}{gr}$$

And for a spherical body of diameter d and density δ:

$$F = \frac{v^2 \pi \gamma}{6gr} d^3 \delta = Ad^3 \delta$$

For two different bodies of diameter d_1 and d_2 and densities δ_1 and δ_2 :

$$\frac{F_1}{F_2} = \frac{d_1^3\ \delta_1}{d_2^3\ \delta_2}$$

If the two bodies are of the same size then :

$$\frac{F_1}{F_2} = \frac{\delta_1}{\delta_2} = n$$

That is for bodies of the same size centrifugal force directly varies with the density.

In the case of bodies of equal density, that is $\delta_1 = \delta_2$:

$$\frac{F_1}{F_2} = n^3 \text{ and } F_1 = n^3\ F_2$$

That is to say centrifugal force varies as the cube of the diameter.

For equally falling bodies :

$$d_1\ (\delta_1 - 1) = d_2\ (\delta_2 - 1)$$

and when $\delta_1 > \delta_2$ then it can be shewn that :

$$d_1^3\ \delta_1 < d_2^3\ \delta_2$$
$$\therefore\ F_1 < F_2$$

It therefore follows that of two equally falling particles the denser has the less centrifugal force.

For two equally falling bodies let $\delta_1 = n\ \delta_2$

Then $$F_1 = n \left(\frac{\delta_2 - 1}{n\delta_2 - 1}\right)^3 F_2$$

Comparing the centrifugal force of equal falling particles of quartz, iron pyrites and galena with densities, $\delta_2 = 2.6$; $\delta^1 = 5.2$ and $\delta_1 = 7.5$

we get for iron pyrites $F_1 = .111\ F_2$
galena $F_1 = .039\ F_2$

Therefore the centrifugal force of iron pyrites is 11 per cent and that of galena 4 per cent more than that of quartz.

(b) The movement of Particles *en masse.*

As in certain dressing operations for example in jigging, grains do not fall freely but in the interstices between their neighbours Prof. *H.S.* Munroe experimented on the fall of particles through tubes and *en masse.* (See The English *versus* the Continental System of Jigging. Trans.: Am.: Inst.: Min.: Eng. Vol. XVII p. 637).

The results he obtained are as follows:

With D representing the diameter of a tube, and the other nomenclature as before:

$$v = 5.11 \left(1 - \left(\frac{d}{D}\right)^{\frac{3}{2}}\right) \sqrt{d(\delta - 1)}$$

The more nearly $d = D$ the slower the rate of fall.

The falling velocity is but little affected if:

$$d < \frac{D}{10}$$

The maximum falling velocity is when:

$$d = \frac{4}{10} D$$

For spheres moving *en masse.*

$$v = .833 \sqrt{d(\delta - 1)}$$

Spheres *en masse* have therefore only about $\frac{1}{6}$ of the velocity of free falling spheres.

An increase in the velocity of a current beneath a mass of shot lifts them and tends to force them apart, the interstitial space or D increases and a greater velocity is required to support the shot. The same material may therefore be worked in jigs run at different speeds and with different strokes.

In rounded grains of uniform size falling *en masse:*

$$v = .490 \sqrt{d(\delta - 1)}$$

For angular grains of uniform size falling *en masse:*

$$v = .536 \sqrt{d(\delta - 1)}$$

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For large spheres moving in a mass of small spheres when the difference in diameter is considerable :

$$v = .307\sqrt{d(\delta - 1)}$$

In a mass of grains of different sizes the large grains move relatively in smaller channels than the small grains and are therefore more retarded.

If the diameter of a quartz grain is d and the diameter of a galena grain d_1 :

Then $$.307\sqrt{d,(2.6 - 1)} = .833\sqrt{d_1(7.5 - 1)}$$

or the diameter of *equal falling grains* are in the ratio $\frac{d}{d_1} = \frac{31}{1}$ and not in the ratio $\frac{4}{1}$ which only applies to free falling grains.

For separation, grains of quartz and galena might be sized between limits of say 1 and 30 mm.

From the above formulæ, velocity of jig currents may be calculated, and from this the length and number of strokes.

From Prof. Munroe's investigation unless the difference in specific gravity is small, close sizing is not necessary, the small stuff in a mixture being concentrated in the interstices between the coarse stuff forming the mineral bed.

For fine stuff on jigs, close sizing is a disadvantage.

The size of the mesh of the jig sieve is of great importance. It determines the size of the particles in the jig bed and these determine the size of the interstices.

Stuff less than $\frac{4}{10}$ the size of the smallest interstices between the coarse stuff, can not be treated successfully.

It has already been shewn that the greater the density of a particle the greater is the space described during the short interval of time while there is acceleration. The short duration of the strokes of a jig are therefore favourable to the separation of unsized material and therefore the size ratio of particles is even greater than that which has been deduced.

CONCENTRATION OF COARSE MATERIAL.

Jigging.—The operation of jigging may be illustrated by taking a mixture of materials like galena and calcite in particles of about the same size, and placing them in a sieve which is rapidly moved up and down in a vessel of water. At each downward movement the galena particles fall more quickly than the calcite, the result after a series of movements being that the galena forms a layer on the bottom of the sieve and the calcite remains on the top. In this case the jig acts as an hydraulic sorter.

It has been stated that when the strokes are short, a jig may be used as an hydraulic sizer.

Comparing for instance quartz and galena we see from the table (p. 243) that a spherical grain of quartz 4 mm. in diameter falls through water at about the same rate as a similarly shaped grain of galena 1 mm. in diameter, although the former is 64 times larger and twenty two times heavier than the latter. It would therefore appear that to separate different materials by jigging, the particles ought to be approximately of equal sizes, and as the relative speeds of round, oblong and flat grains when falling in water are roughly as 1.12 : .97 : .79 they ought as far as possible to be of similar shapes.

In the English system of jigging there is no preliminary sizing. The crushed ore is first treated on coarse or *roughing* jigs and that which passes through these is then treated on fine or finishing jigs—A layer of coarse material is kept on the jigs. In the Continental system, the materials are first sized and the different sizes treated on different jigs.

The two systems may be combined.

In the English system the plant is simple and requires less power to run. It is good for low grade ores on a large scale. The chief objection is the imperfect concentration of material that has passed the jig bed. *Hutch work* may be retreated on finishing jigs, or classified in pointed boxes and retreated in jigs and the slime on tables.

The materials which are subjected to jigging are usually the *smalls* from crushing between $1\frac{1}{3}$ and $\frac{1}{25}$ inch in diameter. It has only been

in exceptional instances that materials of 2 m. and even as low as 1 mm. in diameter have been jigged.

The smallest size that can be treated satisfactory may be taken at 1.5 mm.

These are often too rich to be thrown away and too poor to be directly smelted. If they were reduced by fine crushing for concentration, considerable loss might be incurred by the floating away of fine particles.

Before jigging, according to the continental method the particles require sizing, different sizes being treated separately. The upward movement of the water or the downward movement of the sieve must be quick. Successive movements must not succeed each other too quickly or the particles have not time to settle. With very fine materials the time of settling will be greater than with the coarse materials. The size of sieves corresponds with that of the grains. If ore is finely disseminated in the gangue, it is not jigged.

The laws governing the jigging of fine materials like sand and coarse meal classified as equally falling particles by Spitzkasten or Spitzluttee, are those relating to the period of acceleration (see p. 246 and 247).

Fine jigging was tried at Clausthal for slimes 1 and 5 mm. but the treatment by central discharge tables gave better results. At Scharley in Upper Silesia sand jigs have replaced round buddles. It would however seem that fine jigging can only be resorted to in special cases.

If the piston be provided with valves, and valves be arranged below as in a pump, the water may be caused only to move upwards and during the down stroke the ore is settling in comparatively still water to form a more regular layer.

The piston lift H is calculated from the formula:

$$H = \frac{146.4\sqrt{D(\delta - 1)}}{n\,\pi}$$

D = the sieve class (diameter of hole in sieve).

δ = density of the particles.

h = the number of lifts per minute.

As a test of the accuracy of Prof. Munroes' invesigations, the mill of the St. Joseph Lead Co. Bonne Terre, Mo.: is given. The roughing jigs treat unsized stuff from 4 mm. to slime.

Pistons make 150 strokes of 2 inches per minute. Piston area is $\frac{1}{3}$ sieve area.

The velocity of the current is therefore .08 m. per second which ought to raise a 4 mm. grain of galena. Jig sieves are No. 6 with 2.8 m. openings. The grain of maximum velocity is about $\frac{1}{8}$ mm.

Finishing jigs treat stuff 1 mm. and less. There are 270 stroke of $\frac{1}{4}$ inch per minute. Jig sieve is No. 8 (2 mm).

The concentration is satisfactory, the stuff treated being first from 2 mm. to $\frac{1}{8}$ mm. and afterwards 1 mm. to $\frac{1}{20}$ mm. without loss in tailings.

Movable Jiggers.—The simplest form of a jigger is a sieve which is jerked up and down beneath the surface of water contained in a *tub*, *kieve* or *hutch*. The movement may be given directly by hand, or the sieve may be suspended from a lever which is worked by hand or by a cam.

The sieve may be circular, square or oblong, about 6 or 9 inches deep and 4 or 8 feet area. Stuff which passes through the meshes of one jigger may be treated on a second jigger with a finer mesh.

For 1 square foot area, 4 cubic feet of material may be jigged per hour.

Stationary Jiggers.—These jiggers consist of a sieve or set of sieves arranged in a box filled with water, which by the reciprocating movements of a piston, is caused to pulsate up and down in the sieves. The movement causing the *smalls* to rise must be quick. The piston stroke is 4 or 5 inches and there may be 40 to 80 strokes per minute.

If stuff is fine so that water does not readily penetrate between its component parts, it has to be worked in thin layers. For very fine materials less than 2 mm. in diameter, there may be 120 to 150 strokes of $\frac{1}{3}$ to $\frac{2}{3}$ inch per minute.

Continual Jiggers.—In *Rittinger's Setzherd* the sieves are supported on a frame which is caused to swing through a range of 2 to

4 inches, when it comes in contact with a block and receives a jerk. This takes place during the downward motion of the piston.

The result is that the piston causes a vertical separation of the grains of ore, which, in consequence of the blows, are travelling from the upper end of the sieve, where they are fed from a hopper, towards the lower end where they are sized and dischaged. With a sieve 13 inches broad, 30 to 40 cubic feet may be discharged per hour.

In some continual jiggers, the rich material is discharged through an opening in the bottom of the sieve by the opening of a valve which is lifted at each downward stroke of the piston.

In *Kaiser's patent jigger* as used as Mount Bischoff, the chief point in the construction is the fastening down of the sieve to prevent warping.

The jigging machine most commonly in use is the one where the piston works in a box at the side of the sieves.

One jig may have 2, 3 or 4 sieves, the lighter materials from the top of one sieve being carried by a current of water upon the second sieve, where the jigging action is continued.

Jig boxes are usually made of wood strengthened with angle iron. The piston may have an area equal to that of the sieve. The width of the sieve is 18 to 22 inches and length 28 to 36 inches. The piston has about $\frac{1}{2}$ inch play round its edges. The sieves are made of perforated sheet iron or brass wire.

Length of stroke.

For 2	inches	stuff	=	$5\frac{1}{4}$ inches.
„ $1\frac{1}{2}$	„	„	=	$3\frac{1}{2}$ „
„ 1	„	„	=	$2\text{-}2\frac{1}{2}$ „
„	fine	„	=	$\frac{1}{2}\text{-}\frac{1}{3}$ „

The number of strokes for coarse stuff may be 75 per minute, while for fine stuff 150 to 200 per minute.

The thickness of the mineral bed upon the sieve, is between $\frac{4}{5}$ and $3\frac{1}{5}$ inches, the stuff to be separated above this being $1\frac{1}{4}$ to 8 inches thick.

Other jigs are the *Huet* and *Geyler*, *Colloms jig*, the *Patent jig of Kitto and Paul.*

Dry Jigging.—Where it is difficult to obtain water, materials of different specific gravities may be separated by a current of air, heavy

particles being carried to a less distance by a given current than light particles with similar and equal surfaces. Such separation may be carried on in covered airways or chambers.

In *Aufermann's dry jigger* the arrangement is in many respects similar to a wet jigger, the ore resting on a sieve and being repeatedly lifted by a blast of air instead of a pulse of water from beneath. It separates grains less than $\frac{1}{40}$ inch in diameter.

Coal cleaning.—At many mines the coal is cleaned by picking out the dross by hand—the coal being passed over a shaking screen or travelling band which moves about 40 feet per minute. The cost varies from 1½ *d.* to 2 *d.* per ton for every 5 per cent of dross removed. For small coal, as for example that used in coke and briquette making, the first operation is sizing on trommels, after which it is jigged. The cost may be from 2 *d.* to 6 *d.* per ton. Amongst Machines we have the rotary form as used in Tokyo (*Robinson type*)—the ordinary jig (*Southgate Engineering C°*)—the *Bell and Ramsay* trough machine &c.

In a complete plant, jigs, inclined trommels, elevators, mixers, conveyors, and for briquette making, rolls and pitch breakers (edge rolls) might be required.

CONCENTRATION OF FINE MATERIAL.

1. Sorting into equally falling grains.

Pointed Boxes.—(Spitzkasten). These boxes are in form square pyramids with the base upwards. They are arranged in series, the smallest box being at the upper end and the largest and deepest at the lower end. They are connected by gently inclined troughs. A current of water flowing downwards carries the sands. In the first box the heavier and larger grains are deposited. The current being less swift in the second box another quality of equally sinking grains remain. In the last box the finest material is caught.

The accumulated sands are drawn off by pipes from the bottoms of the boxes.

The top box has one tenth of a foot width for each cubic foot of material received per minute. The breadths of a series of boxes are in the ratio of 1. 2. 4. 8.

If the first box has a length of 6 feet, the other boxes are each 9,12 ,15 feet long.

The sides have an inclination of 50°.

The inclination of the conveying trough per 6 feet is :

For coarse sand		1 to 1½	inch.
„	middle fine	½ to ¾	„
„	„ „	¼ to ½	„
„	„ slime	⅛ to ¼	„

The percentages received from the boxes are :—

1st.	box	40	per cent	sand.
2nd.	„	28	„ „	„
3rd.	„	18	„ „	„
4th.	„	10	„ „	„

At Schemnitz, Hungary, where 20 tons are crushed in 24 hours, four boxes have the following dimensions

1	box.	6	feet	long	2¾	feet	wide	4	feet	deep.
2	„	9	„	„	5	„	„	6	„	„
3	„	12	„	„	8	„	„	8	„	„
4	„	15	„	„	15	„	„	10	„	„

At least 10 heads of stamps are required for a set of boxes.

The size of the discharge orifice is regulated by a nozzle.

If a set of *spitzkasten* has the same dimensions, to obtain classification, the outlets are made of decreasing diameters and syphon discharge pipes are used, those dischanging the coarse material being the shortest.

Spitzlutte.—Here there is a series of inverted wedge shaped boxes, inside which are smaller wedge shaped boxes which can be raised or lowered. The sands flow in the conduits formed between the two boxes, therefore first descending towards the edge of the wedge and then ascending. The wedge shaped box is raised or lowered according to the quantity of stuff that may be flowing. The discharge of the coarser material is from the bottom of the outside box, which may be 3 feet deep. The width of the conduit between two wedges is 3 to 6 inches. *Spitzlutte* are adapted to classify a small volume of thick slime, or to sort several classes of sand and coarse meal from a volume of dilute slime, this latter being separated in *spitzkasten.*

With *spitzlutte* or *spitzkasten* classifiers when sorting sands, an upward current of clear water is admitted from the apex of the wedge. In the former apparatus this increases the upward current, while in the latter it creates one. In a set of classifiers for fine sands the upward current of the first may be 6 inches per second, the factor of diminishing velocity in succeeding classifiers of larger size being from .35 to .50. The sectional area of the current, diameter of discharge outlet are more easily regulated in the spitzlutte than in spitzkasten. (*Kunhardt*).

In some forms of classifiers the pulp is conveyed by a pipe to the bottom of the pointed box where it meets with the upward current of water.—In others it may be admitted to the upward current half way down the pointed box. One box may work 6 to 10 tons of pulp per 24 hours.

Troughs, strips, launders, runs, canals, labyrinths.—Slimes and sands may be separated by deposition, as they flow along a series of troughs. The first troughs along which the materials flow are the narrowest and most steeply inclined, and in them the current is most rapid. For 1 cubic foot per minute, a breadth of 6 inches may be given. The breadths of a series of troughs are in a geometrical ratio, the highest exponent being 1.5, or in the ratio, 1, 1.5, 2.25, 3.37. Thus to separate 8 cubic feet per minute; the breadths of the troughs might be 4, 6, 9 and 13 feet, and the lengths 12, 18, 24 and 30 feet. The inclination of the upper troughs about $\frac{1}{2}$ to $\frac{3}{4}$ in. per 6 feet. The lower ones are horizontal. The troughs may be in series or in broken lines according to space. Sometimes they are placed in duplicate series so that one set may be in use while the other set is being cleaned.

The quantity of material deposited in the various troughs may be,

1st	trough	60	per	cent.
2	„	12	„	„
3	„	10	„	„
4	„	10	„	„
loss	„	8	„	„

The percentage of metal in each division may be the same.

Some of the loss from the *strips* may be saved by deposition in *slime pits*, through which the water is allowed to flow before finally escaping from the works.

One source of loss, is due to the partial drying of the mineral in the strips before it is fed upon *slime tables*, resulting in the production of float mineral.

Blanket Tables.—These are troughs about 10 or 15 feet long, 15 or 16 inches broad, inclined at about 10° or 15° and covered with coarse blanket. They are extensively used at quartz mills in Australia and California for catching the gold as it flows from the batteries. Several troughs are used side by side, so that while the blankets of one are being washed, the gold sand may be flowing over the others. If there is a series of troughs extending longitudingly 20 or 60 feet in length, the upper blankets may be washed every hour, while the lower ones may only require to be washed once a day. The upper blankets may contain 70 or 75 per cent of the gold.

The gold obtained is mixed with iron pyrites and other material and it is therefore further washed by hand in pans.

Blanket stuff may also be concentrated on tables or be directly amalgamated.

2. Classification of equally falling grains. (Concentration on tables.)

Theory of separation.—From pointed boxes and other apparatus described, equally falling grains are obtained. These may be separated by the action of a *thin* stream of water on an inclined table, the water acting with more force on the larger grains than on the smaller grains.

The separation depends upon:—

1. The velocity of flow which may be regulated by the inclination of the table.
2. The thickness of the stream water.
3. The consistency of the sands or slimes.

Slimes and sands may be conducted from pointed boxes directly to the tables, while material from troughs requires to be first diluted with water and supplied by a feeder.

Stationary Feeders.—One type of feeder is a box 3 or 4 feet long, 1½ foot broad and 1½ foot high, with a base inclined forwards and sloping right and left towards the sides. A stream of water is so

arranged that it is admitted at the upper end of the bottom, and as it flows down beneath the sands, it spreads right and left in towards the sides. The stuff as if comes out is diluted with water.

Rotating feeders.—In an ordinary rotating feeder there is a circular conically shaped disc furnished with radiating ribs of iron. As this slowly rotates, the ribs of iron pass beneath and close to the base of one or more hoppers filled with sands, and each rib carries with it a definite quantity of material which is eventually washed off the rotating cone by a jet of water.

The diameter of the cone	= 45 inches
Inclination „ „ „	= 3 in. per foot
No. of ribs	= 48
Revolutions per hour	= 6 to 10
Work per hour	= 4 to 10 cubic feet.

Hand Buddle.—This is a box 12 to 14 feet long, 22 inches wide and deep. The inclination for 12 feet may be from 20 to 6 inches. The latter is for very fine material. The sands are charged about $\frac{1}{2}$ a cubic foot at a time at the head of the box, and as they tend to travel downwards are, raked upwards. When the buddle is nearly filled, say in 1 hour, the deposit is divided into 3 parts,—*head*, *middle* and *tail*. These three parts are washed separately in other buddles. It will be observed that as the material accumulates, the inclination of the washing surface decreases and the *head* continually tends to become poorer.

Sleeping Tables.—Are similar but wider than hand buddles. They are used for fine material.

The Knife, Propeller or **Impeller Buddle.**—This is an inclined plane down which the sands flow. Close above this is a revolving cylinder with projecting paddles arranged like a screw. As these slowly rotate the lighter materials are forced off the side of the inclined plane while the heaviest come out at the end.

Sweeping Tables.—Length 20 to 30 feet, 3 to 4 feet wide and sides 6 inches high. They are used for finer materials than hand buddles.

When the table is covered with stuff, the slime is washed off, after which the concentrated ore at the upper end of the table is washed through a slit which is opened in the middle of the table.

Inclination for sands 10 to 12°

„ slime 5 to 6°

The diluted stuff that can be washed per minute is:

Sands .3 to .5 cubic feet containing from 6 to 10 lbs. sand.

Slimes .08 to .12 „ „ „ „ .28 to .78 „ „

The slimes ought to flow over the table at a low velocity. If they move at a high velocity the particles may slide rather than *roll* and be carried off with the tailings.

Convex Buddle.—This may be described as a circular box 16 or 20 feet in diameter with a gently inclined conical bottom and sides 15 or 30 in hight. In the center there is a more steeply inclined cone about 6 feet in diameter. The slimes and fine materials for which these buddles are used is supplied at the center. It then flows down the inner cone and drops about 1 foot upon the outer more gently inclined cone. As it travels towards the periphery, its surface is gently swept by brushes hanging from revolving arms, which are attached to a central vertical shaft.

The arms revolve 3 to 6 times per minute. In 2 or 3 hours the buddle may be filled, the heavier materials remaining near the center and the lighter round the periphery. The materials may then be divided into *head*, *middles* or *fore and hind krazers* and *tails*, and each product rewashed. A disadvantage is that as materials accumulate the inclination changes. In Cornwall, ore with 2½ to 3 per cent cassiterite is dressed up to 93 to 94 per cent, the loss in tailings being from 3 to 6 lbs of tin ore per ton. It is said that slimes with 1 lb. of tin to the ton may be washed with profit.

Concave Buddle.—Here the base is inclined inwards at about 6°, the central portion of the cone being a circular box about 6 feet in diameter. Four revolving arms carry scrapers or brushes to sweep the slimes as they flow inwards. Here there is a larger area for the accumulation of the heads, and the slimes are more readily carried off as the velocity of the inflowing stuff increases as it approaches the center. The speed of the arms varies with the stuff.

Sands 8 revolutions per minute.
Fine slimes 14 to 16 „ „ „

Amongst other concave buddles there are *Borlase's* where the central outflow may be adjusted, and *Munday's*, so largely used in Australia.

Linkenbach's tables.—These tables are coned surfaces of iron covered with cement and supported on iron frames. The supply pipes, distributing aprons at the apex of these cones which are placed one above the other, and the wash pipes, revolve. Single tables have been made 26 feet in diameter. At Ems the apparatus concentrates lead ore up to 38 per cent, but it has worked up to 65 per cent.

Percussion Tables.—Percussion tables are tables suspended by rods or chains so that they are free to swing. By means of a cam they are pushed forwards and then suddenly released, when they swing back and strike a fixed block. The result of a succession of these blows, which may be given at the end or side of the table, is to cause the heavier particles of ore to travel towards the side or end of the table where the blow is struck, more rapidly than the lighter particles.

End blow Percussion Table.—Length of the table 12 feet. Breadth 5 feet. Height of sides 11 inches at upper end; 5 inches at lower end. Length of chains and rods by which it is suspended at upper end 4 feet, at the lower end 6 feet. Beneath the table is the percussion rod which strikes against a fixed percussion block.

By raising or lowering the chains at the lower end, the inclination of the table may be altered; while by altering the point of attachment to the table of the upper chains the effectiveness of the percussion may be varied. When the percussion rod rests against the percussion block the chains are not perpendicular. By the movement of the cams the deviation of the chains from the perpendicular is increased. The stuff enters and is distributed over the head of the table where the blows are struck. As the heavier materials accumulate at this end, the chains at the lower end may be raised and the slope of the washing surface kept constant.

A table 5 feet wide requires per minute.
Diluted sands .5 to .7 cubic feet;
Diluted slimes .10 to .17 cubic feet.

One cubic foot of water may contain,—Sands 20 to 40 lbs., slime 5 to 10 lbs.

No: of strokes per minute,—for sands with elastic percussion blocks 12 to 16 with rigid 40 to 50,—slimes with rigid block 60 to 80.

Stroke. For sands 12 inches, for slimes 4.8 inches.

One table will wash per day of 10½ working hours from 16 cwt to 1 ton of stuff. The total quantity of water required is .83 cubic feet per minute. It does not give satisfactory results if the slimes are tough. The tables are used for low grade slimes. The slimes which flow off the table may be retreated from 3 to 12 times. One table can treat per day (22 hours) 30 tons of coarse meal, 5 or 6 tons of pulp. As the headings are retreated at least twice, the capacity per day becomes 8 to 10 tons of ore slime and 1½ to 2 tons for pulp. The deposit is allowed to accumulate until it is 6 to 8 inches thick and the time required for this is 3 hours for ore sands, and 9 to 12 hours for pulp.

The Lühriq Compound Vanner, consists of several end blow tables the surfaces being travelling bands so arranged that the *middles* from upper tables are automatically fed upon the lower tables. It treats 9 tons per 10 hours.

The *Dodge Concentrator* is an end blow table where the mineral moves up the inclined concentrating surface, whilst the light worthless material is swept downwards.

Halley's Concentrator is an end blow table where the heavy mineral is caught in a depression of the washing surface.

Rittinger's side blow table.—In this table the blows are given on one of the long sides. By means of a series of revolving cams the table is pushed sideways against a long wooden spring. As each cam passes, the table is suddenly released, and it then flies back to strike the percussion block. The stuff is admitted at the upper end of the table on the opposite side to that on which the blow is struck. The lighter and larger particles tend to travel down the table in almost a straight line, while the denser particles travel downwards more diagonally, being finally discharged at the bottom of the table on the side of the percussion block. The materials are therefore discharged continuously.

Length of table 8 feet; breadth 4 feet.

Two tables each of 4 feet wide are built on one frame.

The inclination varies between 6 and 3 degrees, the latter inclination being for fine materials.

Quantity of stuff supplied per minute. Sands 22 cubic feet, containing per cubic foot 17 lbs. of dry stuff. Slimes 10 cubic feet, containing per cubic foot 6.8 lbs. of dry stuff.

In 24 hours, one double table treats,—sands 5.38 tons, slimes 1.08 tons.

Clear water required per foot width. Sands 22 cubic feet. Slimes 13 cubic feet.

Strokes. Sands 70 to 80 per minute each $2\frac{1}{2}$ inches, slimes 90 to 100 per minute each $\frac{1}{2}$ to $\frac{3}{4}$ inches,—tough slimes 120 to 140 per minute.

The wooden spring is 11 feet long, 3 inches wide, $2\frac{1}{2}$ in thick.

Four double percussion tables concentrate 10 to $15\frac{1}{2}$ tons in 24 hrs.

The loss is about 20 per cent.

.25 *H.P.* is required per table per hour.

On four double tables 10 to 15 tons may be concentrated in 24 hrs.

The washing surface is usnally a fine grained wood, but surfaces of iron, or slabs of marble give good results. The framing is of wood, iron or steel. If two tables moving in opposite directions are fixed to the same foundation the tendency to move laterally is neutralized. If material is too coarse, say over 5 mm. so that it readily rolls, or if it is too fine, say less than .25 mm. like certain pulp, it can not be satisfactorily treated on side blow tables. These tables give rich headings.

For *pulp* slime the *Frue Vanner* or *Embrey Concentrator*, is better than side blow tables.

Rotary Buddles.—These buddles are employed for the concentration of fine sands and slimes.

Rittingers Concave Buddle.—In this buddle an inverted cone or flat 16 faced pyramid, which is slowly rotated by a vertical shaft, forms the washing table. The stuff to be washed is admitted at 4 or 8 points round the periphery. As the table turns, the stuff passes water distributors, which wash the ore as it flows down towards the center of

the table. Still farther and just in advance of the next ore distributor each compartment comes beneath a strong jet of water which cleans the section of the table over which it flows. At the lower end of each section of the table there are holes which carry the various products into different gutters and launders beneath the table.

Diameter of table	16 feet
Inner diameter	7 feet
Length of washing slope	$4\frac{1}{2}$ feet
Inclination	6° to 9°
Revolutions per hour	4 to 6

Stuff passing each distributing board 8 to 12 inch wide per minute,

Cubic foot of sands	.35
„ „ „ slimes	.15

Quantity of dry stuff per cubic foot,

Sands	10 lbs.
Slimes	5 lbs.

Loss 20 to 25 per cent. Ten to fifteen buddles require 1 *H.P.*

Water required per minute,

Sands	15 cubic feet
Slimes	8 „ „

Work per 24 hrs,

Sands	6 to 8 tons.
Slimes	2.8 to 3.6 „

Convex rotary buddle.—These buddles are made from 10 to 18 feet in diameter. The inclination is about 5° One revolution in $2\frac{1}{2}$ to 5 min.

There is only one charge for a complete revolution, when the heavier materials are washed away by a strong jet of water.

They perform their work more quickly and better than ordinary sleeping tables.

One half to three quarters *H.P.* per apparatus. Of an ore easily dressed. 22.5 tons dry weight of coarse meal or 6 tons of fine pulp may be washed in 22 hours. The grain of the wood on the table surface is at right angles to the direction of flow.

Good results may be obtained by mounting a concave buddle or central discharge table above a convex buddle or outward discharge table.

On the upper table coarse *midlings* will be deposited, the remainder (fine pulp) being washed on the lower table. Such an arrangement has given satisfactory results at Clausthal. Fine pulp with 2½ per cent of low grade argentiferous galena has been profitably washed on round tables.

Brunton's Table.—In this machine there is a table 10 feet long and 4 feet wide furnished at the ends with rollers carrying a cloth. which beneath the table hangs down and dips in a tank of water. The stuff is distributed on the cloth and the light particles flow down its length. The heavier particles however are carried upwards against the direction of the stream and over the end of the table down to the tank of water where they are deposited.

Frue Vanner.—In this machine a continuous rubber belt 4 feet wide and 27½ feet long passes over light rollers to form the surface of an inclined plane on which the stuff is concentrated. Below, the belt which is flanged, passes beneath rollers into a box containing water where the concentrates are deposited. While the belt travels the table is shaken back and forth laterally.

The belt travels up hill at from 3 to 12 feet per minute at the same time receiving about 180 to 200 back and forth side shakings. The inclination of the belt is from 4 to 12 in. per 12 feet.

The surface of the belt lasts 2 or 3 months when it is painted with rubber paint The belt receives ore from a distributor about 4 feet from its upper end. As it travels upwards, it meets with small jets of water from a clear water distributor.

The water required is 3 to 6 gallons per minute. One machine treats 6 tons per 24 hours of stuff passing a 40 mesh screen, or 2 Vanners to 5 stamps.

Materials may be treated directly from the batteries and 1 man can attend to 16 machines. Each machine requires one quarter *H.P.*

It does not appear to be well adapted to obtain *middlings*. In gold washing where the mineral is often reduced by the stamps to a finer condition than the gangue, from 80 to 85 per cent may be saved A saving of 90 per cent has been attained.

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Embrey Concentrator.—This differs from the Frue Vanner in that the shaking is longitudinal instead of lateral.

The Triumph Concentrator is very like the *Embrey.*

Golden Gate Sulphuret Concentrator.—This consists of a tray 11 feet in length which has a longitudinally reciprocating movement. Material fed on the tray is caused to travel along it to a "protecting plate" where the inclination of the tray changes. The heavy material passes beneath this plate, and the light material above it is drawn off by an exhaust pipe.

Rittinger's Separirtrichter.—In this machine where advantage is taken of centrifugal fore, a wing wheel revolves in a vessel 2½ feet in diameter and 5½ feet high. The water contained in this vessel being in revolution, the stuff to be separated is admitted above. As it descends to the bottom, a portion of the stuff passes through a funnel, whilst that which has been thrown the farthest outwards, descends between the funnel and the sides of the containing vessel.

The falling ore may also be collected and discharged from radially arranged compartments at the bottom of the cylinder.

Hundt's Rotating cylinder. (*Stromsetz* machine).—In this machine there is a tub divided radially at the bottom into a series of compartments. In the center of this, revolving on a perpendicularly placed shaft there is a double cylinder, the inner part of which is slightly conical, with its base downwards. As the shaft revolves the cylinders revolve, and with them the water in the tub. Ore is introduced into the space between the cylinders, and as it falls is separated. In consequence of the heavy material falling more perpendicularly than the lighter material which describes a flatter spiral, the general result is that materials of different specific gravities reach different compartments at the bottom of the tub, from which the material as it accumulates is from time to time discharged. The tub is about 5 feet in diameter and the radial compartments 17 inches high. The space between the rotating cylinders at their base is about 2 inches. With ⅜ inch grains, a five feet water column requires 3 revolutions per minute. Coal requires 2 revolutions and 1½ foot of water.

The speed and height of the water column vary with the size of the grains.

In another of Hundt's concentrators, the stuff descends two screw shaped inclined planes which are broader at the bottom than at the top and which revolve on a shaft.

Dolly Tub.—The dolly tub or tossing kieve is a tub about $3\frac{1}{2}$ feet diameter. Water is introduced, and then with a *stirrer* or *dolly* set into a circular motion. While this is in progress, finely divided concentrated ore like tin or lead is introduced until the mixture is thick, after which the stirrer is withdrawn and the mixture allowed to settle. During the settling the outside of the tub is struck with a hammer and the ore is *packed*. The rich heavy ore accumulates on the bottom and the slime above.

The mixing takes about 15 minutes and the settling 20 minutes. It is used for headings from buddles, about 2 cwt being treated at a time.

In the *Duncan concentrator* the material is separated in a pan making about 8 revolutions per minute the pan at the same time having a jigging motion. Three pans are required for 10 stamps. At the Ashiwo mines it gives good results with copper ore.

The Hendy concentrator is a circular iron basin, the bottom slightly convex with a circular trough round the periphery. The basin is caused to turn in a quick vibratory manner on its center.

The heavy materials are moved towards the trough where they are discharged, while the tailings are discharged at the center.

Separation by means of a dense fluid.—Rittinger points out that by means of a solution of zinc sulphate (S G. 1.5), slate (S G. 2.6) might be separated from coal (S G. 1.3) inasmuch as the coal would float and the slate would sink. In separating the constituents of rock masses a solution of iodides of potassium and mercury S G. 3.19 may be employed. Methods of this description on account of the expense of the solution can only be employed upon a small scale.

Under this head Rittinger includes the separation of gold which when pure has a S G. 19.33 from associated minerals by means of mercury S G. 13.6. In this case the separation is chiefly due to the gold amalgamating with the mercury from which it is subsequently separated by distillation.

Information about amalgamation may be found under the section relating to stamps, grinding and tables.

Magnetic Separators.—Magnetic iron sand may be concentrated by passing it over tables or belts beneath or above which, magnets are placed (*Conkling separator*); by passing the sand over or under magnetic rolls (*Bachanan and Wenstrom's Separator*); or by allowing the sand to fall in front of the pole of a fixed magnet, when the magnetic sand will be deflected and the ordinary sand will fall vertically (*Edison Separator*). In the latter the magnet is 6 feet long, 30 inches wide, 10 inches thick and wighs 3400 lbs. It is excited by a current of 16 amperes with an E.M.F. of 116.5 volts, consuming $2\frac{1}{2}$ *H.P.*

For other separators, (see Trans.: Am.: Inst.: Min.: Voll XVII. p. 728. also Vol XIX. p. 65. and p. 187).

Apparatus for Raising Material, &c.

Raff wheels.—For raising slimes and sands, wheels are constructed on the principle of an Egyptian wheel. The diameter of the wheel depends upon the height to which the material is to be raised. The breadth of the wheels may be 6 to 9 in. At distances of about 1 foot round the periphery spade shaped buckets are arranged. These buckets branch from the periphery, at an angle of about 30°. Below, the wheel dips into a trough containing the material to be raised, which is scooped up and discharged at the upper side of the wheel.

Raff wheels are often employed in connection whith crushing by rollers. The material after passing through the rollers is received in a trommel where it is sized. The larger particles not passing through the trommel are received in the buckets of a raff wheel and carried upwards to be again passed through the rolls.

Centrifugal pumps are sometimes employed for slimes.

Jacob's ladders may be used for various materials while for conveying material horizontally or up an incline of less than 30° an endless band or an archimedian screw may be employed.

Treatment of amalgam.—The gold amalgam is collected and strained through a cloth and pressed into balls about $1\frac{1}{2}$ inches diameter. The gold contained is about 33 per cent. The squeezing may be done in a screw press. The balls are then placed in a cast iron retort and the mercury driven off by heat; time occupied being 2 or 3 hours. The loss of mercury is 3 per cent.

To determine weight of sand or slime contained in stuff from boxes, batteries etc.

1. A given measure of the stuff is weighed, after which the water is drained off and the sands or slime dried, when it is again weighed. The difference between these weights is the weight of solid material in the given quantity of stuff.

2. The density of the dry sand may be determined by observing how much water is displaced by a given weight of the sand when introduced into an ordinary specific gravity bottle.

4. Rittinger recommends that a graduated glass tube, each division of which corresponds to .025 cubic inch of water be filled with water to a division n, after this p grains of the dry sand are put in and the water rises to the division n_1. The density δ is then given by the formula.

$$\delta = \frac{1598\, p}{n_1 - n}$$

4. Another method recommended by Rittinger to find the weight P and the density δ of the sand, is as follows.

Let a 1/2 or 3/4 litre bottle be half filled with the wet sand and then filled up with water. The weight of this $= P_1$

Remove the water and then fill up to the same mark with a solution of zinc sulphate (S G. 1.5) The weight of this $= P_2$

Now if v = volume, δ the density of the sand.
v_1 = „ $\delta_1 = 1$ „ water.
v_2 = „ δ_2 = „ zinc sulphate.

Then $P_2 = v\,\delta\,\gamma + v_2\,\delta_2\,\gamma$
$P_1 = v\,d\,\gamma + v_1\,\gamma$
$Q_1 = (v + v_1)\,\gamma$
$v_2 = v_1$
$P = v\,\delta\,\gamma$

$$P = \frac{P_1\,\delta_2 - P_2}{\delta_2 - 1}$$

$$\delta = \frac{P_1\,\delta_2 - P_2}{P_1 + Q_1\,\delta_2 - (P_2 + Q_2)}$$

MACHINERY REQUIRED IN MILLS (*Fraser and Chalmers*).

Ten Stamp Wet Crushing Silver Mill.

1 No. 2 Blake Crusher 10 × 7 inches.
1 Grizzley or ore screen 4 feet × 10 feet.
2 Tulloch automatic ore feeders.
10 Stamps of 850 lbs. each in one battery including all iron work, wooden pulley and hard wood guides for stamp stems.
1 Set of water pipes for battery.
6 Combination amalgamating pans 5 feet diameter complete.
3 Combination settlers 8 feet diameter complete.
1 Clean-up pan, 4 feet diameter, complete.
1 Set of water pipes with hose for pans and settlers.
1 Set of steam pipes from boiler to pans.
1 China pump to return surplus water from settling tanks to supply reservoir.
1 Automatic quicksilver system complete, including quicksilver elevator, distributing tank, bowls and pipes.
3 Amalgam safes.
1 Amalgam car.
1 Retort 12 inch 4 feet complete with smoke stack.
1 Sheet-irom plate for retort-room floor.
1 Bullion melting furnace with crucible, tongs and bullion mould.
1 Overhead crawl and one ton differential pulley block with chain to be placed over stamps.
1 Do. of two ton capacity over pans.

All track iron and nails for crawls.

1 Line of shafting running under pans and coupled to engine, with bearings and pulleys to drive pans and settlers.
1 Countershaft for stamps with bearings and pulleys.
1 Countershaft for crusher with bearings and pulleys.

All belting and lace leather.

1 Corliss engine 14 × 42 inches.
1 Tubular boiler 45 × 16 inches complete.
1 No. 3 steam feed pump.
1 Tubular heater 20 inches diameter.

All pipe connections to make power complete.

The capacity of the above mill averages about 25 tons of ore each 24 hours.

The steam power described is of capacity for the 10 stamp mill only as described. It is advisable to put in an engine capable of driving a 20 stamp mill necessitating a Corliss engine 18 × 42 inches and when the additional 10 stamps with pans and settlers, etc., are added, a second 54 × 16 boiler must then be introduced.

Ten Stamp Dry Crushing Silver Mill,

1 No. 2 Blake crusher 10 × 7 inches.
1 Grizzley or ore screen 4 feet × 10 feet.
3 Tulloch automatic ore feeders (one for ore dryer).
1 Automatic revolving ore drying cylinder 44 × 36 inches × 18 feet complete.

10 Stamps of 850 lbs. each in one battery, including all iron work, wooden pulleys and hard wood guide boxes for stamp stems.

1 Screw conveyor for front and back of stamps to convey pulp to elevator.

1 Elevator with belt, elevator cups, pulleys and shafts.
1 Screw conveyor, overhead from elevator to furnace hopper.
1 Roasting furnace, either Brückner, Howell, White, O'Harra or Stetefeldt.

1 Hot ore car and track iron with nails.
4 Combination amalgamating pans 5 feet dia.
2 Combination amalgamating settlers 8 feet dia.
1 Clean-up pan 4 feet dia.
2 Set of water and steam pipes for pans and settlers,
1 Line shaft running under pans and coupled to engine with bearings and pulleys to drive pans and settlers.

1 Countershaft for stamps with bearings and pulleys.
1 Countershaft for crusher and ore dryer with bearings and pulleys.

1 Countershaft for roasting furnace with bearing and pulleys.

All belting and lace leather.

1 Amalgam retort 12 inches × 4 feet, complete, with smoke stack.

1 Plate for retort room floor.
1 Bullion melting furnace with crucible tongs and bullion mould.
1 Overhead crawl and one ton differential pulley block with chain to be placed over stamps.
1 Do. of two ton capacity to be placed over pans.
1 Corliss engine 14 × 42 inches.
1 Tubular boiler 54 inches × 16 feet, complete.
1 Steam feed pump No. 3.
1 Tubular heater 20 inches dia.

All pipes, valves and fittings to connect engine, boiler, pump and heater to make power complete.

The capacity of the above mill averages 15 tons each 24 hours.

The steam power described is of capacity for the 10 stamp mill only as described. It is advisable to put in an engine capable of driving a 20 stamp mill neccessitating a Corliss engine 18 × 42 inches and when the additional 10 stamps with pans and settlers, etc, are added, a second 54 inches × 16 feet boiler must be added.

Ten Stamp Gold Mill,

1 No. 2 Blake crusher 10 × 7 inches.
1 Grizzley or ore screen 3 feet × 10 feet.
2 Tulloch automatic ore feeders.
10 Stamps of 850 lbs, each in one battery, including all iron work, wooden pulley and hard wood guides for stamp stems.
1 Set of water pipes for battery.
2 Copper table plates 54 inches × 8 feet × 1 to 8 inches thick, pure L.S. copper free from flaws.
4 copper lining plates for mortars 3 to 16 inches thick, same quality.
1 Amalgam retort and condenser.
1 Countershaft for stamps with bearings and pulleys.
1 Countershaft for crusher with bearings and pulleys.

All necessary belting and lace leather.

1 Engine 9 × 14 inches 20 *H.P.*
1 Boiler 40 inches × 10 feet, complete.

1 Feed pump with belt.
1 Heater.
All pipe connections to make power complete.

The capacity of this mill will average 25 tons of ore each 24 hours.

Power required for a 10-Stamp Wet-Crushing Gold Mill.

1	"Blake" rock-breaker, No. 2	= 6	horse power.
2	Ore-feeders	= 0	„ „
10	Stamps, 750 lbs., 90 drops	=12	„ „
4	Frue Vanner concentrators	= 2	„ „
1	Grinding pan. 3 feet diameter	= 3	„ „
1	Settler	= 3	„ „
Friction		= 4	„ „
	Total	=30	„ „

The above form of mill is capable working 15 to 18 tons per day of twenty four hours.

Power required for a 20-Stamp Wet-Crushing Gold Mill.

1	"Blake" rock-breaker, No. 2	= 6	horse power.
4	Ore-feeders	= 0	„ „
20	Stamps, 750 lbs., 90 drops	=23	„ „
8	Frue Vanner concentrators	= 4	„ „
1	Grinding pan, 3 feet diameter	= 3	„ „
1	Settler	= 3	„ „
Friction		= 7	„ „
	Total	=46	„ „

The above is capable of working 35 to 40 tons of ore per day of twenty-four hours.

Power required for a 10-Stamp Wet-Crushing Silver Mill.

1 "Blake" rock-breaker, No. 2	=	6	horse power.
2 Ore-feeders	=	0	„ „
10 Stamps, 750 lbs., 90 drops	=	12	„ „
6 Grinding pans, 5 feet diameter	=	30	„ „
3 Settlers, 8 feet diameter	=	9	„ „
Friction	=	7	„ „
Total	=	64	„ „

The above power is capable of working 18 to 20 tons of ore per day of twenty-four hours.

Power required for a 20-Stamp Wet-Crushing Silver Mill.

1 "Blake" rock-breaker, No. 2	=	6	horse power.
4 Ore-feeders	=	0	„ „
20 Stamps, 750 lbs., 90 drops	=	23	„ „
12 Grinding pans, feet diameter	=	60	„ „
6 Settlers, 8 feet diameter	=	18	„ „
Friction	=	13	„ „
Total	=	120	„ „

The above power is capable of working 40 tons of ore per day of twenty-four hours.

Allowance is made for grinding in the pans in both of the above cases.

Power required for a 10-Stamp Dry-Crushing Silver Mill.

1 "Blake" rock-breaker, No. 2	=	6	horse power.
2 Ore-feeders	=	0	„ „
10 Stamps, 750 lbs., 90 drops	=	12	„ „
1 "Howell White" furnace, 40 inches	=	4	„ „
4 Amalgamating pans, 5 feet diameter	=	8	„ „
2 Settlers, 8 fcet diameter	=	6	„ „
Friction	=	9	„ „
Total	=	45	„ „

Water required for a Quartz Mill.

The quantity of water required to work either gold or silver ores by wet battery process is generally estimated as follows:

For boiler, 7½ gallons per horse power per hour.
For each stamp, 72 gallons per hour.
For each pan, 120 gallons per hour.
For each settler, 60 gallons per hour.

If the water used in the battery, pans and settlers be run into settling tanks it can be re-used with a loss of about 25 per cent.

Ten-Stamp Free Ore Silver Mill.

(*Sixth annual Report of the State Mineralogist. California* 1886)

One 4 feet by 12 feet grizzly.
One 8 by 10 Blake crusher.
Two automatic ore feeders.
One 10-stamp battery, 750 to 800 pound stamps.
Four 5-foot combination amalgamating pans.
Two 8-foot settlers.
One 3 foot clean-up pan.
One amalgam safe and strainer.
One quicksilver elevator, with tanks, pipes, etc.
Two traveling crabs for battery and pans.
One 14 inch silver retort.
One melting furnace.
Shafting, pulleys, boxes, etc., for mill.
Belting for mill.
Pipes and fittings complete.
One Duncan concentrator for saving quicksilver and amalgam.
Weight of the above, 87,000 pounds.

Power required:

One 50-horse power engine.
One 50-horse power boiler.
One 50-horse power feed-water heater.
One No. 3 steam pump.
Steam and water connections.
Weight of the above, 24,500 pounds.
Total weight of the above, 111,500 pounds.

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Twenty-Stamp Free Ore Silver Mill.

One 4 feet by 12 feet grizzly.
One 8 by 10 Blake crusher.
Four automatic ore feeders.
Two 10 stamp batteries.
Eight 5-foot combination pans.
Four 8-foot settlers.
One 4-foot clean up pan.
One quicksilver elevator, with tanks, pipes, etc.
Three traveling crabs and track for batteries and pans.
Two amalgam safes and strainers.
Two 14-inch silver retorts.
One melting furnace.
Shafting, pulleys, boxes, etc., for entire mill.
Belting for entire mill.
Pipes and fittings for mill.
One Duncan concentrator, for saving amalgam and quicksilver.
Weight of the above, 180,000 pounds.

Power required:
One 100-horse power engine.
Two 50-horse power boilers.
One 80-horse power feed water heater.
One No. 4 steam feed pump.
Steam and water connections.
Weight of the above, 50,000 pounds.
Total weight of the above, 230,000 pounds.
Grizzly, 4 feet by 12 feet, weighs about 2,000 pounds.
8 by 10 Blake crusher weighs about 8,000 pounds.
Automatic feeder weighs about 700 pounds.
5-foot pan, combination, weighs about 6,200 pounds.
8-foot settler weighs about 6,000 pounds.

Ten-Stamp Gold Mill.

One grizzly, 4 feet by 10 feet.
One 8 by 10 Blake crusher.
Two automatic ore feeders.
One 10-stamp battery, 750 pound to 800 pound stamps,

One traveling crab for battery.
Sixty square feet of silver-plated copper plate, No. 14 guage, plated 1 ounce of silver to the square foot.
Shafting for mill.
Battery, pipes, and fittings.
Belting for mill.
One gold retort.
Weight of the above, 37,500 pounds.

Power required:
One 25-horse power engine.
One 30-horse power boiler.
One 20 horse power feed water heater.
One No. 2 steam pump.
Steam and water connections.
One No. 2 steam pump.
Steam and water connections.
Weight of the above, 10,500 pounds.
Total weight of the above, 48,000 pounds.

Twenty-Stamp Gold Mill.

One grizzly, 4 feet by 12 feet.
One 8 by 10 Blake crusher.
Four automatic ore feeders.
Two 10-stamp batteries, 750 pound to 800 pound stamps.
Two traveling crabs for batteries.
One hunded and twenty square feet silver-plated copper plates, No. 14 gauge, plated 1 onnce of silver to the square foot.
Shafting for mill.
Battery pipes and fittings.
Belting for mill.
One gold retort.
Weight of the above, 72,000 pounds.

Power required:
One 50-horse power engine.
One 50-horse power boiler.
One 50-horse power feed water heater.
One No. 3 steam pump.

Steam and water connections.
Weight of the above, 24,500 pounds.
Total weight of the above, 96,500 pounds.

Specifications of Twenty-Stamp Gold Quartz Mill for the Buchanan Mine, Tuolumne county, California. (*J. Hamilton.*)

One (1) Grizzly, to be made of $\frac{3}{4}$ inch by $2\frac{1}{2}$ inch wrought iron, 12 feet long, 4 feet wide; spaced 2 inches between the bars, and secured by three $\frac{3}{4}$ inch bolts, and cast-iron thimbles and washers; also end bars.

Rock Breaker.—One Blake crusher, fitted with two flywheels, 4 feet in diameter, and one (1) pulley, 30 inches in diameter, and 12 inch face crank-shaft, to have outboard bearings; holding-down bolts for each, to be long enough for 14 inch timbers; all caps for boxes to be fitted to use zinc oil cups, and "Albany" lubricating compound.

Battery.—Four (4) single discharge gold mortars, each weighing about 5,000 pounds; the bottoms to be $6\frac{1}{2}$ inches thick; well planed; the screw frame bearings to be planed also; the bases to be 26 inches wide, 3 inches thick, and cored for eight (8) mortar bolts $1\frac{1}{2}$ inches in diameter; the feed mouth to be 26 inches long and $3\frac{1}{2}$ inches wide, at the smallest part; the mortars to be well finished, and to be 56 inches long, and about 52 inches high.

Mortar Bolts.—Thirty-two (32) mortar bolts, $1\frac{1}{2}$ inches in diameter, and 36 inches long; nuts on both ends, and thirty-two (32) 4 inch square countersunk washers for the lower ends; one (1) wrought-iron wrench; jaw designed for the nuts of mortar bolts; arm to be 6 feet long, slightly curved, made of 1 inch by $2\frac{1}{2}$ inch iron.

Stamp Dies.—Stamp dies, 7 inches deep, $8\frac{1}{2}$ inches in diameter, with square bases, having beveled corners, and all made to properly fit the mortars, and to be cast of the best car-wheel iron; stamps to weigh 850 pounds each.

Shoes.—Twenty (20) stamp shoes, $8\frac{1}{2}$ inches in diameter, $7\frac{1}{2}$ inches long; to be cast of the best car-wheel iron.

Heads.—Twenty (20) stamp heads, $8\frac{1}{2}$ inches in diameter, 18 inches long; keyholes to be $1\frac{1}{8}$ inches by 3 inches; one (1) key for driving out shoes and stems, and to be made as may be directed.

Tappets.—Twenty (20) double-faced gib tappets, made of the best steel faces, 9 inches in diameter; tappets to be 12 inches long; bands or flanges turned, and each to have gibs and three (3) steel keys, and all to be marked; keys to be boxed for shipment.

Cams.—Twenty (20) double-armed cams, made of best steel; ten (10) to be right, and ten (10) to be left-handed; hubs on one side, and to be strongly banded with $\frac{3}{4}$ inch by 2 inches wrought-iron bands, well shrunk on; the faces of the cams to be well smoothed, and to be fitted to $5\frac{1}{2}$ inches cam shafts, and to be properly marked.

Feeders.—Four (4) Hendy Challenge ore feeders, complete with all latest improvements; also, twenty (20) steel keys for same, to be marked and fitted, and properly boxed for shipment; the cams to be about 32 inches long, with $2\frac{1}{2}$ inches face, and struck to give an $8\frac{1}{2}$ inches drop if required.

Corner Boxes.—Six (6) corner boxes, for $5\frac{1}{2}$ inches cam shafts; cored for 1 inch bolts; backs to be planed true, and bearings to have a strip running lengthwise, and the balance of the bearings to be well babbitted and bored: the ends to be faced; the cap to be solid, bored, but not babbitted, unless required, and bored with three (3) $\frac{5}{8}$ inch holes each, for using "Albany" compound.

Cam Shafts.—Two (2) cam shafts, $5\frac{1}{2}$ inches in diameter, and 14 feet 6 inches long, and key-seated between bearings.

Jack Shafts.—Four (4) jack shafts to be 3 inches in diameter and 59 inches long, with eight (8) bearings for same, and to be made of cast-iron.

Stems.—Twenty (20) stamp stems $3\frac{1}{4}$ inches in diameter, 14 feet long; turned and tapered off both ends for heads; made of rolled iron.

Latch-sockets.—Twenty (20) open latch-sockets; all to be well lined with heavy leather.

Guides.—For (4) complete sets of best white oak guides. The lower guides to be 16 inches wide and 4 inches thick, and the upper

to be 14 inches wide and 4 inches thick; bored for $3\frac{1}{4}$ inches stamp stems, and to be 10 inches between centers; all to be 60 inches long.

Cam-shaft Pulleys.—Two (2) wood pulleys, 72 inches in diameter, with 16 inches faces and 6 inches thickness of wood between the flanges, and to be made of the best kiln dried sugar pine; to be turned and well bolted to 40 inches diameter cast-iron sleeves; flanges for $5\frac{1}{2}$ inches cam shaft; the flanges to be faced.

All of the keys for this mill to be made of the best steel.

Overhead Crabs.—One (1) crab for battery having flanged wheels 7 inches in diameter and 2 feet face; the axles to be $1\frac{1}{2}$ inches square and long enough to set the wheels 10 inches between the flanges; the boxes to be $1\frac{1}{4}$ inches thick, and all made of the best wrought iron.

Piping, etc.—All piping, hose, valves, bibs, cocks, ells, tees, unions, etc., to be furnished as per detailed bill. All necessary bolts, belts, nuts, and washers, also copper plate for battery and aprons, to be furnished of proper weight, and as shown on drawings, $\frac{1}{8}$ inch thick, not plated; also, silver-plated copper plate, 20 inches wide and 20 feet long.

Belting.—One (1) main driving belt, to be about 55 feet long, 20 inches wide, and made of the best 5 ply rubber of the "Boston Belting and Packing Co.'s" manufacture, patent stretched.

Battery Belts.—One hundred and ninety (190) feet of the best 5 ply rubber, 16 inches wide.

Rock Breaker Belt.—Ninety (90) feet of 4 ply rubber, 12 inches wide.

Boxes for Shafting.—All boxes to be planed on the bottom and faced on the ends, and to be well babbitted, and the caps to be bored with $\frac{5}{8}$ inch holes, for Tatum & Bowen's lubricating compound. Two (2) steel collars and set-screws for each.

Shafting and Pulleys.—One (1) main battery line shaft to be made in two lengths, 4 inches in diameter, 20 feet long, and $3\frac{1}{2}$ inches diameter, 20 feet long, coupled together with solid coupling. One (1) countershaft for driving rock breaker, 3 inches diameter, 20 feet long;

both shafts fitted with all necessary bearing boxes, and holding down bolts for 14 inches timbers.

Pulleys.—One (1) main driving pulley to be fitted to engine crank shaft at mine, to be 48 inches diameter, 21 inches face.

One (1) pulley for main battery line shaft, to be 72 inches diameter, 21 inches face, 4 inches gauge.

One (1) pulley to drive crusher countershaft, 40 inches diameter, 12½ inches face, 4 inches gauge.

One (1) pulley to drive battery, 34 inches diameter, 16½ inches face, 3½ inches gauge.

One (1) pulley to drive crusher, about 40 inches diameter, 12½ inches face, 3 inches gauge.

One (1) pulley on crusher countershaft, 26 inches diameter, 12½ inches face, 3 inches gauge.

Tighteners.—Two (2) battery tighteners, 16 inches diameter, 20 inches face, with racks, pinions, frames, handwheels, pawls, and pawl plates, complete for stopping and starting the batteries.

One (1) rock breaker tightener complete, as above specified, for stopping or starting the crusher.

All necessary bolts for batteries and buildings, and all necessary belt rivets for securing the belts. All belting used in this mill to be of Boston Belting Co.'s manufacture, best quality and patent stretched.

All castings to be made of good material, smooth and free from blow-holes or other imperfections. All keys to be made to fit and properly marked.

Mortars are to be cast from selected patterns or drawings, which will be furnished.

Should anything be lacking to complete this mill which has not been specified, it must be supplied the same as if fully mentioned.

Everything to be delivered on the ground (at company's expense), and timbers to be cut according to plans to be supplied.

NOTE.—The company already had an engine and boiler. The building, also, over the mill was built by the company. The cost of the above material was $5,985, and of the labor, $2,300. Save a slight

change in the tappets, the work was satisfactorily done, and the mill crushed over two tons per stamp, of hard quartz, every 24 hours.

The total cost of grading, building, machinery, etc., was a little short of $15,000.

Examples of Processes.

The first seventeen of these examples are recommended by Fraser and Chalmers, manufacturers of mining machinery in Chicago.

1. *For free gold that can be panned out and no spulhurets.* Wet stamping and quicksilvered copper plates. In the Black Hill $8 ore yields good returns, the milling only costing 60 cents per ton of ore.

2. *Free gold with sulphurets.* By sulphurets are meant iron and copper pyrites, galena, blende. Tellurides, selenides, antimonides may also occur with these ores together perhaps with silver.

   Wet stamping and catching the products in blankets or hides. Subsequent treatment in a pan or arrastra. A better method is, wet stamping and quicksilvered copper plates. The tailings are treated on a vanning machine (two of Frues vanners to 5 heads of stamps) subsequent roasting, treatment with chlorine gas and lixivation with water, the gold being precipitated with ferrous sulphate. Chlorination costs 8 to 10 $ per ton of concentrates. Another process is to roast and amalgamate in pans.

3. *Free gold in small quantity but with much silver in the sulphurets.*

   Roasting milling or free gold milling with Vanners for the tailings and subsequently smelting. Sulphides foul amalgam. When these are present dry stamping, mixing with salt, roasting to form chlorides and then lixivation. The alternative is to smelt the concentrates.

4. *Chloride of silver, native silver and decomposed portions of silvee vein out crops containing over 6 oz per ton.* Free silver milling.

5. *Silver ores, partly chloride and partly silver bearing sulphurets.* Free milling, Vanners and smelting (see 3). For high grade ore, roasting milling (see 3).

6. *Silver ore with base metal sulphuret if low grade.* Fine concentration and smelting or for higher grade, roasting and milling.

By fine concentration is meant fine crushing and treatment on Frue Vanners, revolving buddles, Evan's slime table, Rittinger's tables, end blow tables or some form of slime machine. The process costs $1 to $2.50 per ton. Smelting cost $8 to $30 per ton.

7. *Low grade silver ores with gray copper, ruby, britile or native silver.* Fine concentration and smelting.

8. *Heavy mineralized ores of lead, copper, zinc often carrying silver.* Fine concentration and smelting.

9. *Highly mineralized ores of lead, copper and zinc.* Fine concentration and smelting.

10. *Carbonate or oxide of lead and copper.* Smelting.

11. *Solid galena.* Hand selection or coarse concentration and smelting. Coarse concentration means, coarse crushing, sizing on revolving screens, jigging and treatment of slimes on a slime machine. In Western camps it costs $1 to $3 per ton.

12. *Metallsc copper ores.* Stamping, coarse concentration and melting to ingot.

13. *Antimony ores.* Hand picking, concentration and smelting for metal.

14. *Zinc blende and zinc carbonate.* Coarse or fine concentration and reduction by zinc smelting process.

15. *Tin ores.* Fine concetration, roasting and smelting.

16. *Copper pyrites, copper glance.* Hand picking, concentration, partial roasting and matting; or if convenient selected and concentrated products are sent to refineries. If of low grade, lixivation for copper and silver.

17. *Heavy iron pyrites carrying gold.* Chlorination process or roasting and intermixing with ores for smelting.

18. *For lead, silver lead, with blende, pyrites &c.* the following installation may be employed.

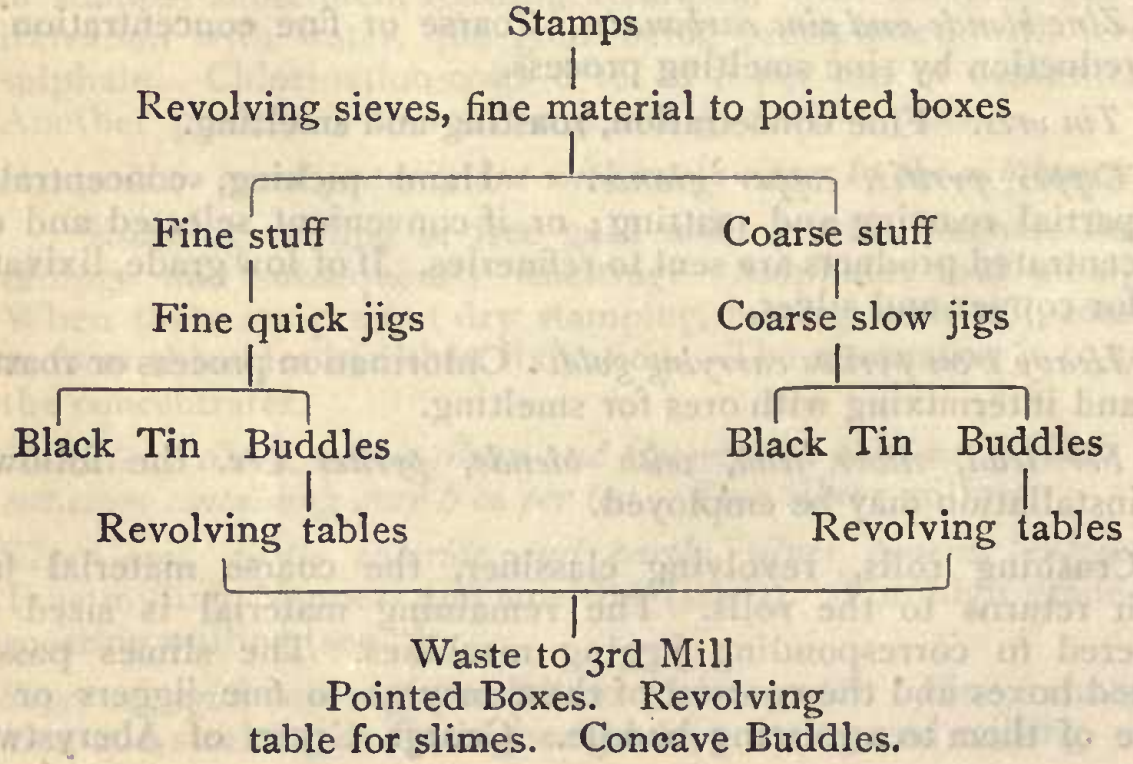
Crushing rolls, revolving classifier, the coarse material from which returns to the rolls. The remaining material is sized and delivered to corresponding jigging machines. The slimes pass to pointed boxes and the coarsest of these may go to fine jiggers or the whole of them to a rotating buddle. George Green of Aberystwyth

recommends for a capacity of 15 tons per day, 3 self acting. jiggers, 3 patent classifiers and 3 round buddles. (With wooden hutches jiggers cost £270, with iron hutches £310). (Rolls 28 inches × 16. inches capacity per day of 35 tons, cost £140).

19. **Port Philip Company, Clunes, Victoria.**—*Ore*,—gold quartz with pyrites. *Machines*,—Two Appleton's rock breakers. Stamps 80 heads, square heads 6 to 8 cwts each. Grates thick copper plates with 84 holes to sq. in., buddles to collect pyrites each 24 feet diameter, 5 amalgamating barrels. Shaking table. Two Chilian Mills for grinding roasted pyrites, one 24 inches cylinder engine 60 *H. P.* working up to 120 *H. P.*

20. **Defiance Mills, Charters Towers, Queensland.**—*Ore*,—gold quartz with pyrites. *Machines*,—Battery 25 heads. Quicksilvered copper plates, shaking table and plates, convex buddle, sands entering at periphery, with Mondy's patent scrapers, Wheeler's pans and Berdan pans.

21. **Mount Bischoff, Tasmania.**—*Ore*,—tin stone in loose soil which is dressed from 1½ or 2½ per cent up to 72 or 75%. The ore is first roughly washed in sluices, the coarse stuff going to three mills. In the first two mills the plan is approximately as follows.



22. **Walnut tree Bridge Colliery.**—By a Jacob's ladder the coal is dropped into a hopper where it meets a stream of water. From the hopper it passes down an inclined plane to an iron jigger. The small coal used for coke comes from the bottom of the jigger while the clean coal passes over the lip.

23. **Minera Lead Mine, Nr. Wrexham.**—*Ore*,—galena with much blende. *Gangue*,—limestone. *Machines*,—a pair of rolls, 4 hand jiggers, 6 or 7 square buddles and one circular buddle. Smalls go to settling pit and fine material is treated in square buddles. Produce 500 tons blende, and 100 tons galena per month.

24. **Van Lead Mine, Wales.**—*Ore*,—galena, with blende, a little copper pyrites and earthy matter. *Treatment*,—picking, Blake's crusher, rolls. The smalls go by a launder to pointed separating box. The fine material goes to concave buddles and the remainder to jiggers. The purest material is washed in square buddles while other ore from the jiggers is washed by hand and the mixed ore is ground fine and then washed on an Impeller buddle. The fines from the separating box go to a conical buddle 18 feet diameter. Produces 100 tons blende, and 200 tons of galena per month. Cost of dressing works £60,000.

25. **Roman Gravel Lead Mine Nr. Shrewsbury.**—*Ore*,—galena, blende with calcite and blue elvan country. *Treatment*,—Spalling, yielding rich, middle and waste. Washing and picking. In the mill we have,

Rolls, two revolving sizing sieves giving materials to two classes of jiggers. The fine materials from the sieves and rolls pass to two sizing boxes with upward currents of water, each of which give material to a set of fine jigs with 11 or 12 holes per inch. The fine slimes passing the sizing boxes pass to a settling box and then to circular buddles. Mixed ore from jigs is recrushed and washed on a special jigger.

The first revolving sieve has 4 holes per inch and the second 5 holes.

Produce about 300 tons of galena per month, dressed to 82 per cent of lead.

26. **Langley Barony Lead Mine** (*Nr. Newcastle*).—The ore is galena with calcite, barytes, a little quartz, earthy matter and very little blende. Here are rolls, trommels, jigs as above. The finest materials is treated on a convex revolving buddle, the clean ore being washed off at the end of each revolution. There are three qualities of dressed ore, with 81, 76 and 73 per cent. of lead respectively.

27. **S. Ives Consols, Cornwall.**—*Ore*,—tin and grey copper. *Treatment*,—Moderate concentration on buddles and frames. Roasting in Brunton's revolving calciner. Washing in square buddles to get rid of copper sulphate from which the copper is precipitated with iron. Tin stuff washed on revolving table. Tossing and packing.

28. **Levant Mine, Cornwall.**—*Ore*,—tin stone and copper pyrites. Produce 100 tons of copper and 12 to 20 tons of concentrated tin stuff per month. *Process*,—picking and the copper ores after rough treatment sold. For tin ore; stamping, buddles 25 feet in diameter (buddling repeated three times) tossing and packing in a kieve. Roasted for 24 hours, the arsenic fumes being caught in a flue. Roasted ore sprinkled with water and copper sulphate removed in solution. Buddling.

29. **Mallenear Copper Mine.**—*Ore*, copper pyrites blende, galena, iron pyrites with fluor spar, quartz calcite &c.

Process,—Ore descends over grates to dressing floor. The top grate of bar iron 4 inches apart, to lower grate of round iron 1 inch apart. Sometimes ores are picked and sized and then repicked. Four classes of ore (A,B,C,D.) are obtained and are treated as follows.

A. Large. Divided into 4 parts.

1st best work, crushed to $\frac{1}{2}$ inch square, sold.

2nd class work, cobbed and separated to waste and best. The latter crushed to $\frac{1}{2}$ inch square, sold.

3rd class work, jigged and waste to heap.

4th class, blende and galena.

B. Middles which do not pass 1 inch holes, go to a water pit and then classified into four divisions as above.

C. Which does not pass ½ inch holes. Washed with hand sieves and classified as above.

D. Which pass ½ inch holes. Hand jigged on sieve, 5 holes per in. The waste scraped from the top. Result gives, *prills* which are sold, and *dradge* with 3rd class from above which goes to machine jig.

Waste or *enemy* rejigged by tributers. Fine from all the above goes to square buddle. The material for machine jigging is put through a pair of 2.′6″ rolls and a revolving sieve 5 holes per inch. After this it is sized by other sieves and the different sizes treated separately in three jigs. Refuse goes to slime pit. Rich from bottom of jigs to setting tanks. Rich from first jig sold, that from 2nd and 3rd. jig is rejigged.

The three jigs have respectively:

1st, strokes ⅝ inch holes per inch 5
2nd, „ ⅝ „ „ „ „ 5
3rd, „ ½ „ „ „ „ 5
} 300 strokes per minute.

Two boys and one man attend to the jigs. Slimes passing 18 holes to the inch are treated on 6 buddles each 20 feet diameter. Two treat the products direct from jigs, and 4 treat the products from the two. Contents of the first two are divided into 3 parts. Heads go to third buddle, middle to fourth, tail and over flow to pits. The contents of the 3rd buddle are divided into three. Heads sold. Middle rebuddled, tail to forth, overflow to pit. Contents of the 4th buddle divided into 3 parts. Heads go to third, middle rebuddled, tails to pit. Settled slimes are treated in 5th and 6th buddles.

Quality of ores

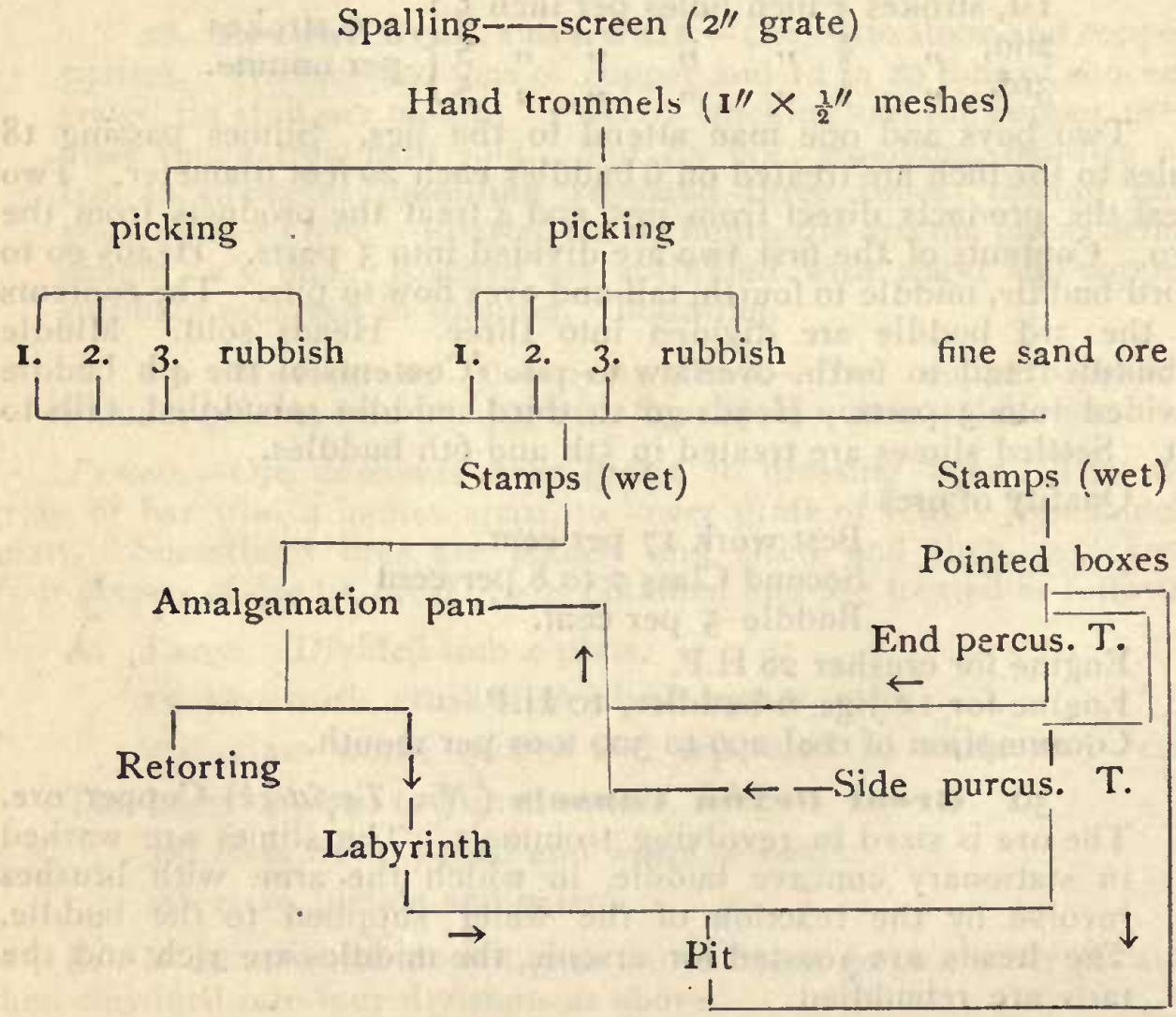
Best work 17 per cent
Second Class 7 to 8 per cent
Buddle 5 per cent.

Engine for crusher 28 H.P.
Engine for 12 Jigs, 6 buddles, 10 H.P.
Consumption of coal 290 to 300 tons per month.

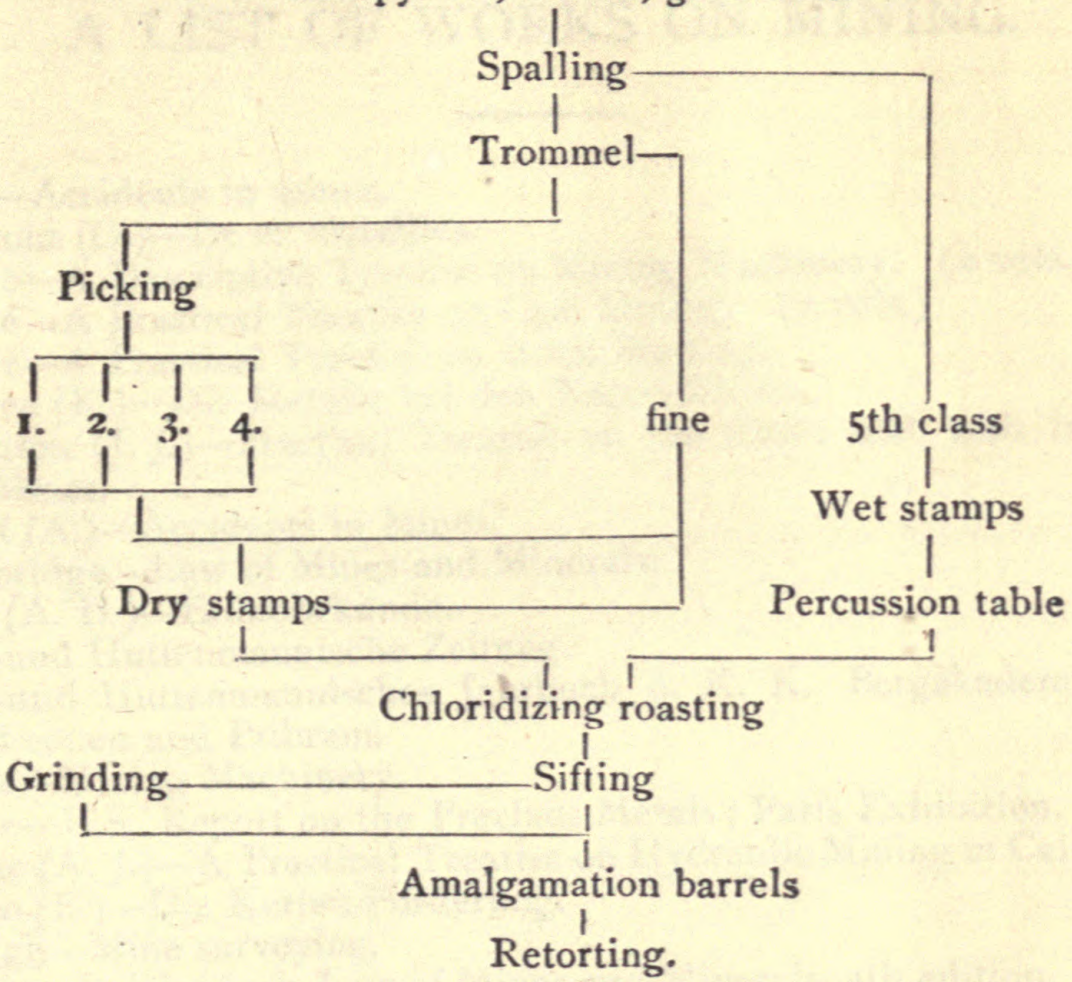
30. **Great Devon Consols** (*Nr. Tavistock*) Copper ore. The ore is sized in revolving trommels. The slimes are washed in stationary concave buddle, in which the arms with brushes revolve by the reaction of the water supplied to the buddle. The heads are roasted for arsenic, the middles are rich and the tails are rebuddled.

31. **General notes on Cornish Tin Mines.**—Where wolfram occurs with tin it is separated by picking, or the reduced and dressed ore is roasted with salt or carbonate of soda and the resulting tungstate of soda subsequently washed out. At East Pool Mine the ore is sold with the wolfram. If tin is disseminated it is crushed fine with stamps in high mortars and no grates, as at Wheal Kitty. This flush system avoids restamping. At Dolcoath and in Camborne generally, the ores are restamped At Tin Crown, convex buddles are used for ordinary work and concave for fine slimes from the frames. In finishing operations many frames are employed.

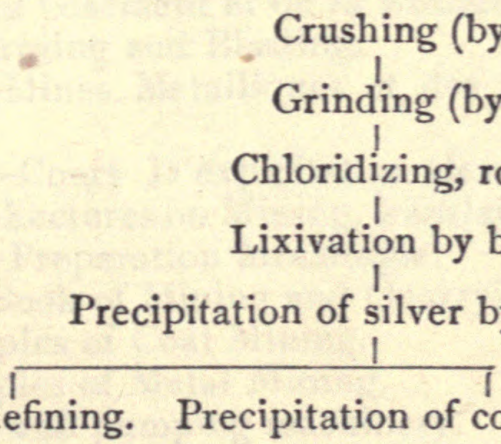
32. **Sado, Japan.**—Ore is finely disseminated silver and gold with sulphurets in quartz. The process was :

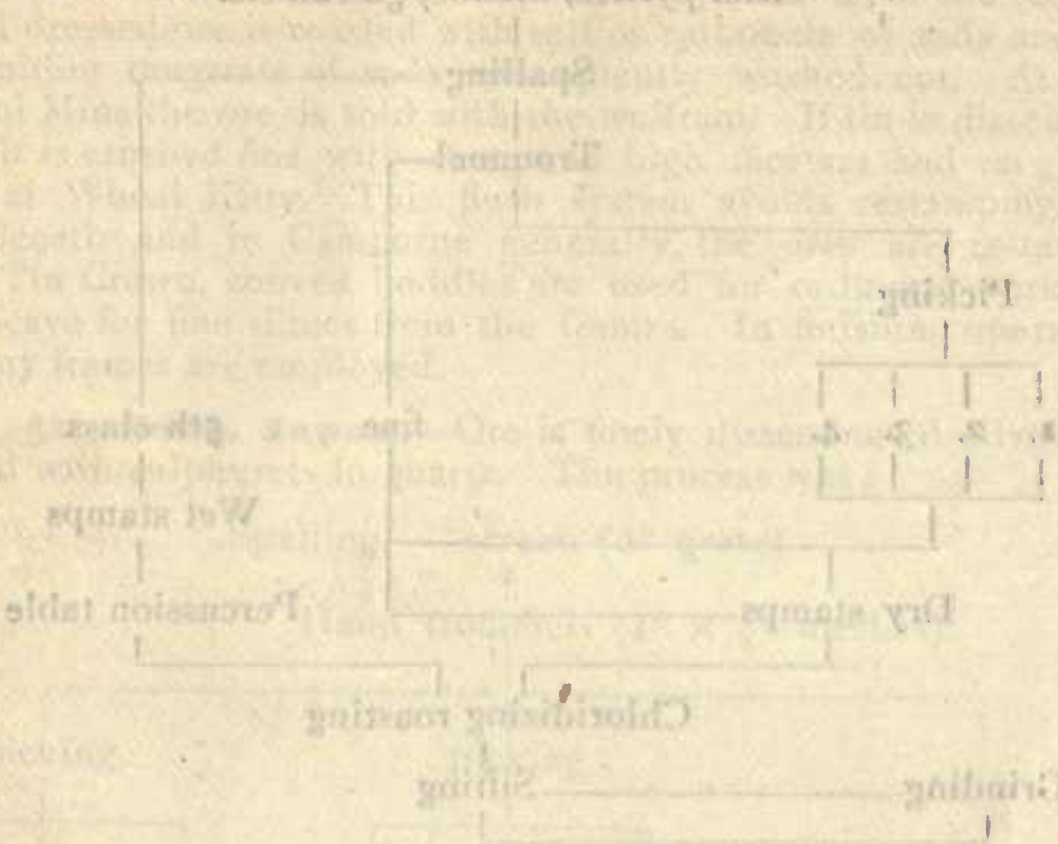


33. **Ikuno Japan.**—Ore is quartz with argentite, copper and iron pyrites, blende, galena &c.



34. **Kosaka, Japan.**—Ore, a yellow clay ore with from 4 to 40 oz of silver per ton and a black sulphuret with blende, copper &c. with about 12 oz of silver per ton.





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